Engineering and Mining Journal

WITH WHICH IS CONSOLIDATED ENGINEERING & MINING WORLD

NEW YORK · LONDON

NOVEMBER
1933

The Story of McIntyre

MODERNIZED GOLD PRODUCTION IN CANADA

McGRAW-HILL PUBLICATION

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Printed by The Schweinler Press, N. Y.

Engineering and Mining Journal

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CONTENTS FOR NOVEMBER 1933

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VOLUME 134 NUMBER 11

The Story of McIntyre

Editorial 445

Gold's Prestige ..... *J. P. Bickel* 446

Growth and Outlook ..... *R. J. Ennis* 447

An Index of Achievement 449

McIntyre Staff Heads and the Features of McIntyre Practice 450

Engineering and Geology ..... *H. G. Skavlem* 451

Development and Mining ..... *D. E. Keeley* 459

McIntyre Metallurgy ..... *J. J. Denny* 472

McIntyre's Equipment ..... *W. F. J. Cosser* 485

Safety and Health ..... *A. D. Campbell* 492

Bibliography 496

Trends and Developments in the Industry ... 497

Market Data 499

McGRAW-HILL PUBLISHING COMPANY, INC.
330 West 42d St., New York, N. Y.
Cable Address: McGrawhill, N. Y.
Chicago: 520 North Michigan Avenue
San Francisco: 883 Mission Street
Montreal: 1188 Phillips Place

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An inquiry to the nearest office may help solve your exploration problem.

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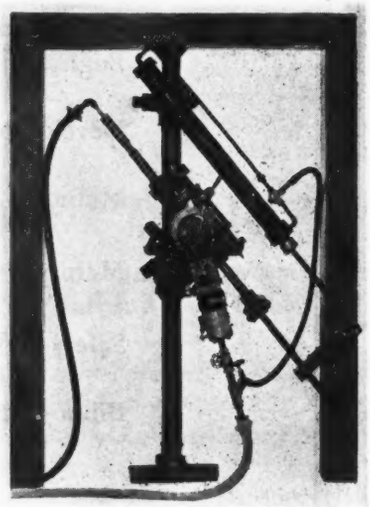

Typical column set-up of No. 10 AIR Light Mitchell Diamond Drill with Air Rod Hoist. Requires only 5½ ft. in line of hole, with 5 ft. rods, and weighs less than 200 lbs.

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Engineering and Mining Journal

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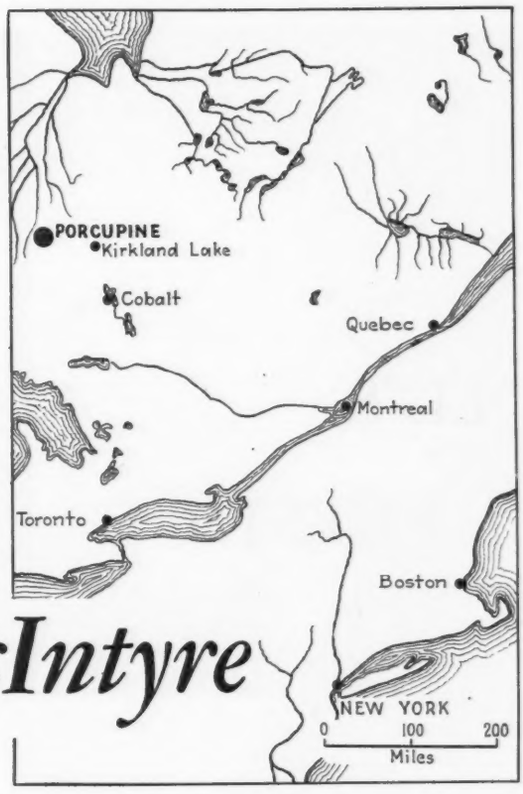
H. C. PARMELEE *Editorial Director*

NOVEMBER, 1933



The Story of McIntyre

CURRENT INTEREST in gold lends an element of timeliness to this issue of *Engineering and Mining Journal,* in which members of the technical staff of McIntyre Porcupine Mines Limited present a notable series of articles describing the company's mining and metallurgical operations. The industry is indebted to the authors for their generous contribution to technical literature. As an example of modernized gold production, the Story of McIntyre will commend itself to mining engineers and metallurgists throughout the world.

No fortunate accident marks the success of a mining enterprise, despite the inherent elements of risk that cannot always be clearly foreseen. A venturesome spirit is essential but not sufficient. It must be supported by technical knowledge and skill of a high order, which, unfortunately, are too often lacking in many an ill-starred mining project.

In the case of McIntyre the factors contributing to the success of the enterprise will be revealed as the articles are read. Technical talent is obviously in command, and a bold spirit of progress and research is in evidence. The reader will be impressed with the fundamental geological investigations and their interpretation, with the metallurgical research and its practical results, with the generous use of modern mechanical and electrical equipment, and finally with the evidence of a lively understanding of the sociological aspects of a modern mining community. In the successful coordination of these factors lies the secret of achieving what is commonly referred to as good management.

It is this kind of talent and management that lends confidence to those who finance mining exploration and development. Not that Nature may not occasionally spring a surprise to confound the best-laid plans, but that the exercise of technical knowledge offers a degree of insurance that should never be lacking. The element of surprise is thus minimized. Dollars invested in mining must obviously be venturesome and patient, but they are entitled to the best safeguards that sound technology and good management can throw about them. It is only thus that the mining industry can gain the confidence and support of the investing public.

The Story of McIntyre exemplifies the venturesome spirit of mining enterprise at its best, with effort crowned by success. The deep development program rewarded the boldness of its sponsors; and the undertaking of still deeper development, calling for large capital expenditure, is evidence of the confidence of the management and the whole-hearted support of the directors.

H. C. Parmelee

GOLD'S PRESTIGE

J. P. BICKELL
President

THE CHAOTIC CONDITIONS prevailing throughout the past two or three years have necessitated re-appraisal in the case of all industries, and from this operation emerges the gold-mining industry with a new and vastly improved prestige. In the hectic days of a year or more ago, some impulsive economic advisers expressed the opinion that gold as a standard of value was no longer possible, and that its future use would be limited to filling teeth and gilding picture frames. More mature consideration, however, has developed a general realization that even though some revision or modification may be deemed necessary, no better standard is available, and that normal conditions of trade and commerce cannot reasonably be expected until the traditional standard is once more restored.

In the meantime, the Canadian gold mines have greatly assisted our governments in meeting their foreign obligations in the terms of the original contract. This ability to discharge such obligations as they fall due has prevented our Canadian dollar from being discounted to perhaps disastrous levels. New equipment and ordinary supplies required by the industry, not only to maintain but to greatly increase the scope of operations, have provided a very welcome market for the sorely harassed manufacturer and primary producer.

In 1931 the average tax paid per ton of gold ore treated in Ontario was 43½c. The comparative figure for 1932 was 59c. With the establishment of a free gold market in London, the world price of gold has moved rapidly upward to a higher level. This means larger profits and therefore heavier taxes. Since tax rates are designed to keep pace with profits, it is estimated that the tax this year will be around 80c. per ton treated, and our contribution to the Provincial and Dominion Governments will be increased accordingly. This increased price for gold is very beneficial to the Canadian people. The revenue resulting from the sale of our product is wholly derived from sources outside of Canada, and plays an important part in the maintenance of our favorable trade balance. If by international agreement the price of gold in terms of currency should be permanently raised, Canada would be assured of many benefits. It would greatly lengthen the profitable life of the gold mines now producing and enable them to increase their present capacity appreciably. It would give a new value to many known prospects hitherto considered too low in grade to permit of economic development. It would result in a great influx of capital from people in foreign countries seeking investment in the mineral industry.

The cumulative effect of such developments would necessitate the employment of a great many additional men, and increase our export of the one product that all countries are anxious to acquire and against which no nation has established import barriers of any kind.

Gold mining in Canada can no longer be termed a questionable venture. It is an essential industry. Our people realize its importance, and undoubtedly everything possible will be done to increase our production and foster the growth of the industry.

McIntyre has endeavored in every way possible to play its full part in the marvelous development of Canada's gold-mining industry.

GROWTH *and* OUTLOOK

R. J. ENNIS
General Manager

McINTYRE PORCUPINE MINES LIMITED received its charter in March, 1911, and shortly afterwards acquired title to some ninety acres of ground originally staked by Sandy McIntyre. It was a very modest beginning, founded largely upon the hope that Hollinger orebodies would extend across the line into the McIntyre ground.

On the whole, surface development produced little of a promising nature. The original underground development down to a depth of say 300 ft. was disappointing. It was only after the auriferous sulphide deposits were opened up in the No. 5 vein that the company was able to make appreciable progress.

Financing prior to this more satisfactory period had been difficult. Heavy liabilities had been contracted, but under the new conditions production soon lightened the load and development was undertaken upon a much more aggressive scale. Before long it was apparent that the orebodies encountered extended into adjoining properties and that ways and means must be found to expand the company's boundaries. The company had no funds available for the purchase at this time, but it did have an enthusiastic and energetic board of directors, who were willing to back such an effort personally to the extent necessary. Under such circumstances the properties of the Pearl Lake Gold Mines and of the Jupiter Mines Limited were purchased upon reasonable terms, and McIntyre's profitable history dates from the time of these acquisitions.

Milling capacity was provided. Gradually the tonnage was increased. Profits permitted the retirement of a bond issue and later the purchase of additional property, previously owned by Plenaurum Gold Mines and Platt Veteran Gold Mines Limited, and with other less important additions the company's property was expanded to a total area of 680 acres.

From a capacity of 600 tons per day in 1916, the company's facilities were gradually increased to about 1,500 tons or better.

In 1924 the directors were advised that if the then rate of production was to be maintained with the necessary development work, additional shaft capacity was immediately required. Inasmuch as the ore was in a general way following the rake of the porphyry at an angle of approximately 45 deg. to the east, underground development had extended away beyond the easy limits of No. 6 or the main operating shaft, and had attained a depth of 2,500 ft.

To make a long story short, the No. 11 shaft was sunk to the 4,000-ft. level. These new facilities speeded development work, and ere long additional capacity was warranted. An addition to the old mill was next considered. The plant was old—partially obsolete. It did not lend itself to expansion and was far from the new shaft. Decision was finally made to centralize operations adjacent to the new shaft, and the present plant was planned and erected.

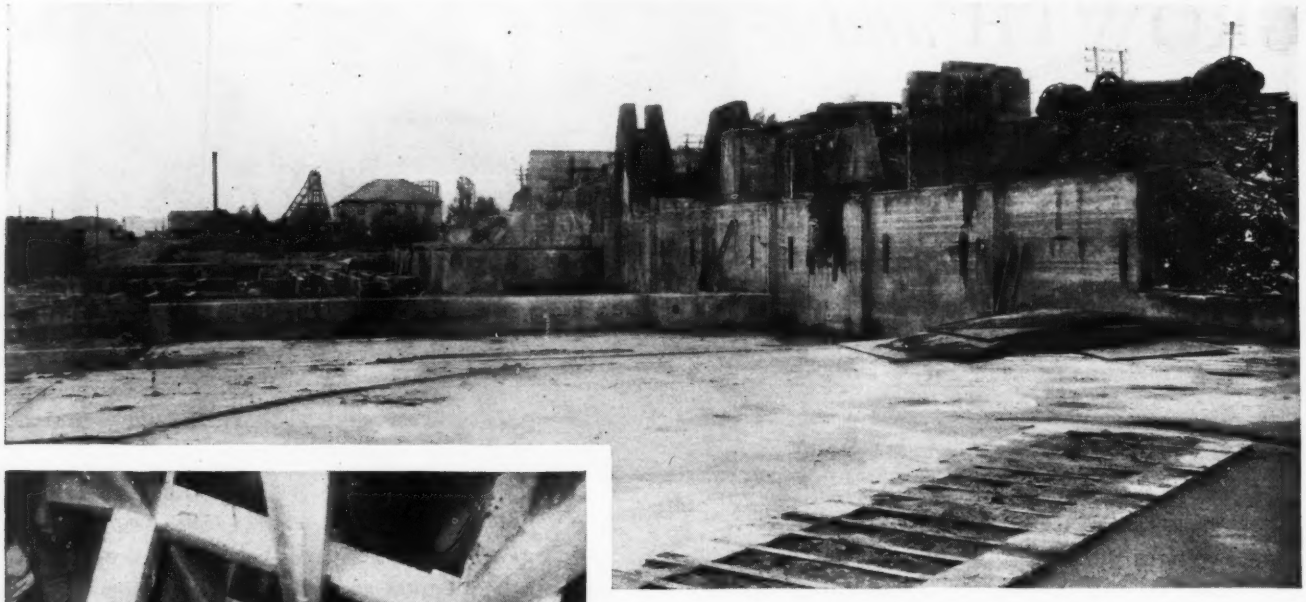
With the different units located in proper relation to one another, and with the advantage of modern improved practice and equipment, the results obtained have fully met our expectations and hopes. Operations are now being conducted at a rate somewhat in excess of 2,000 tons per day.

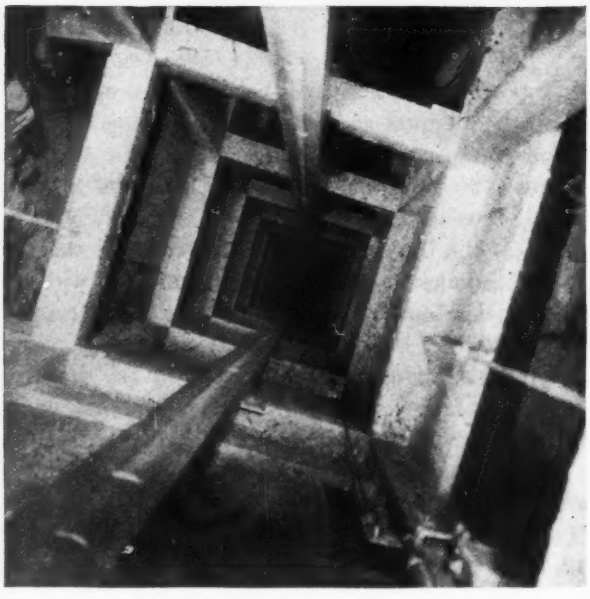
The company's development policy has been consistently aggressive, as is evidenced by its decision to

sink an internal shaft from the 3,875-ft. level and to project development down to the 7,000-ft. horizon.

The staff are a unit in believing that the controlling geological features depend in a large degree upon the attitude of the porphyry stocks. To the extent that development work has proceeded, we know that these stocks have maintained their relative positions at depth. Geological evidence, as we are able to interpret it, supports the opinion that ore will continue to a point that will justify the sinking of the new or deep No. 12 shaft.

The mine never appeared to be in as good condition as it is today, and nothing is discernible on the horizon to disturb our expectations for the future.

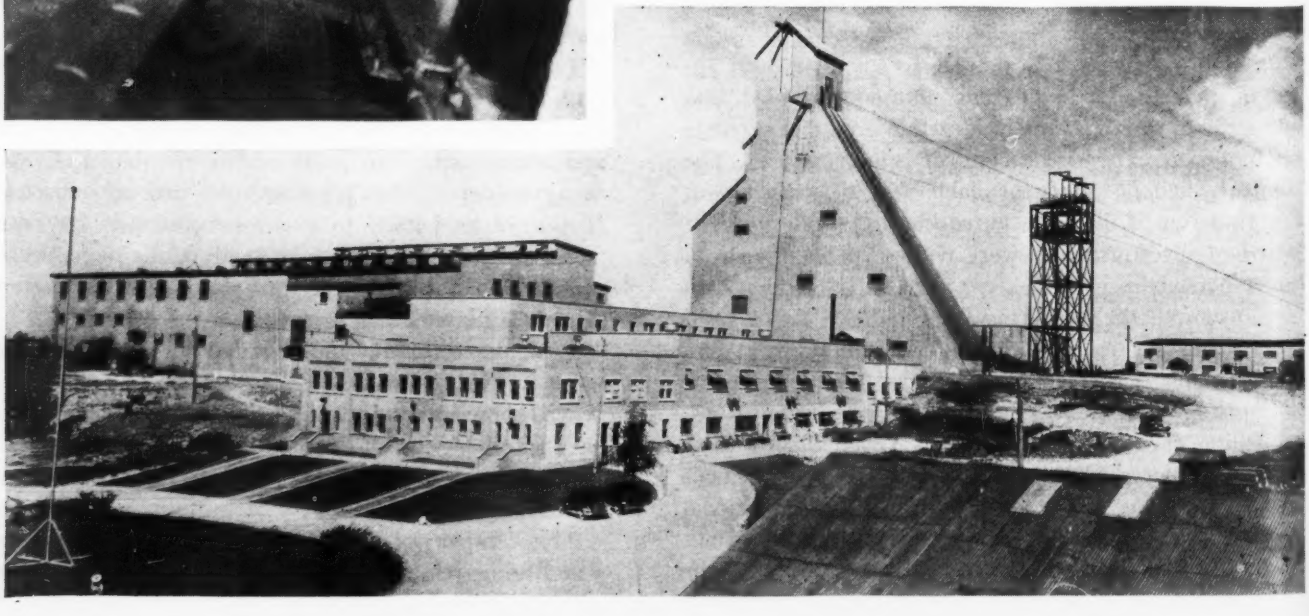




A REMINDER OF YESTERDAY—The site of the old mill, dismantled when the new one was built. Here approximately 86 tons of gold and 17.9 tons of silver were produced in the 18½ years of its existence.

IN QUEST OF A GREATER FUTURE AT DEPTH—Looking down the new No. 12 sub-shaft, on its way to the 7,000-ft. horizon and already sunk a mile below the surface.

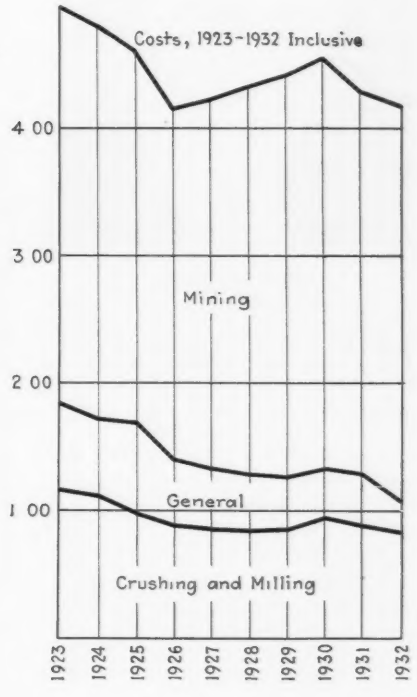
TODAY—McIntyre Porcupine's centralized plant at No. 11 shaft.

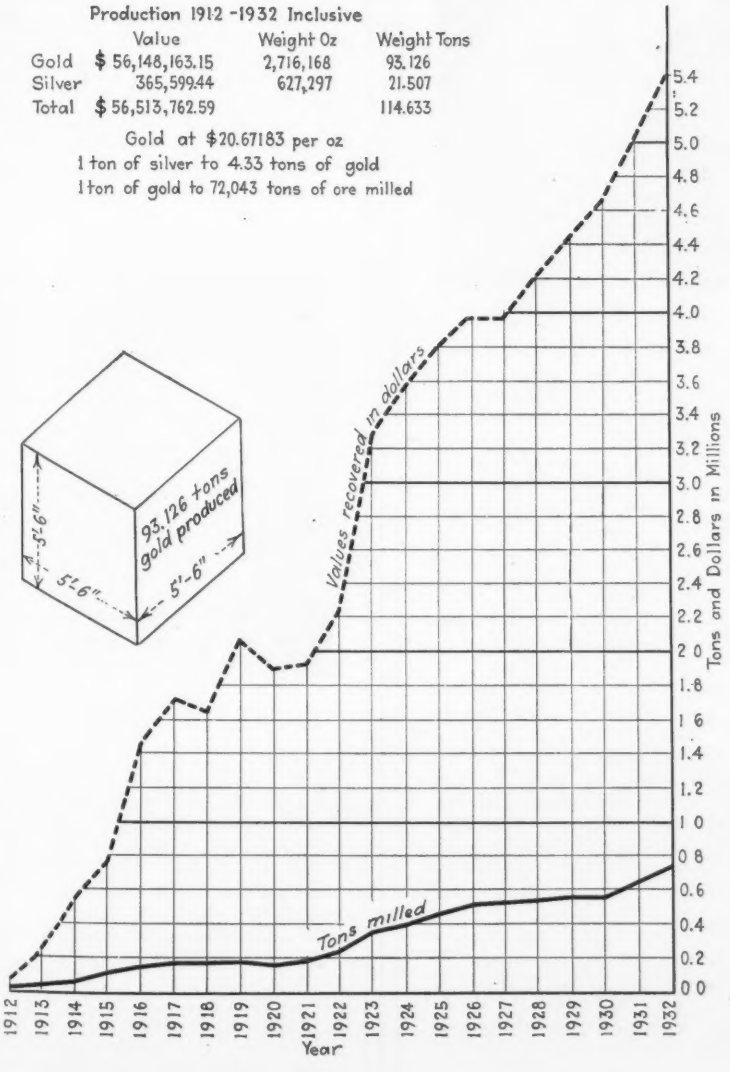


Production of McIntyre Porcupine Mines Limited Since Beginning of Milling Operations in 1912

Date	Period Months	Tons Milled	Value per Ton	Recovery per Ton	Total Value
1912	12	14,500	$7.00	$5.25	$ 76,166.38
1913	12	31,979	7.85	7.05	225,752.25
Jan. 1, '14-Mar. 31, '15	15	85,654	8.87	8.39	718,331.71
Apr. 1, '15-Mar. 31, '16	12	105,758	7.71	7.38	779,990.94
Apr. 1, '16-June 30, '17	15	195,307	10.00	9.55	1,864,914.28
July 1, '17-June 30, '18	12	178,327	10.05	9.61	1,714,258.00
July 1, '18-June 30, '19	12	179,874	9.78	9.29	1,671,646.03
July 1, '19-June 30, '20	12	188,835	11.52	11.02	2,080,178.44
July 1, '20-June 30, '21	12	171,916	11.67	11.08	1,904,326.36
July 1, '21-June 30, '22	12	193,971	10.69	9.99	1,937,105.07
July 1, '22-June 30, '23	12	240,615	9.96	9.35	2,249,741.63
July 1, '23-June 30, '24	12	360,140	$9.69	$9.14	$3,291,178.22
July 1, '24-June 30, '25	12	400,259	9.43	8.86	3,546,637.52
July 1, '25-June 30, '26	12	460,909	8.72	8.25	3,804,774.90
July 1, '26-Mar. 31, '27	9	385,409	8.08	7.67	2,957,060.97
Apr. 1, '27-Mar. 31, '28	12	520,460	8.09	7.66	3,987,634.94
Apr. 1, '28-Mar. 31, '29	12	538,165	8.24	7.83	4,212,624.82
Apr. 1, '29-Mar. 31, '30	12	550,495	8.46	8.05	4,433,626.45
Apr. 1, '30-Mar. 31, '31	12	558,115	8.84	8.30	4,633,140.73
Apr. 1, '31-Mar. 31, '32	12	655,030	8.00	7.63	4,997,461.80
Apr. 1, '32-Mar. 31, '33	12	736,300	7.70	7.37	5,427,211.15
Totals		6,752,018	$8.83	$8.37	$56,513,762.59

AN INDEX OF ACHIEVEMENT





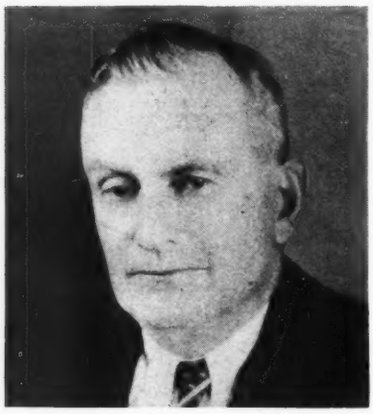
IN ITS RECORD of production and its costs is to be found a measure of the achievement of the McIntyre Porcupine gold-mining enterprise. The conditions under which this adventure has been and is being conducted must be taken into account in any such appraisal. These are described in the pages that follow by the company executives in charge of the various divisions of the work. The facts set forth by these department heads will throw more light on the figures, presented graphically and in tabular form herewith.

McINTYRE STAFF HEADS

and the features of

McINTYRE PRACTICE

as described by them in the following pages



D. E. KEELEY
Mine Superintendent

HIGH LIGHTS of Mr. Keeley's article on "Development and Mining" (pages 459-471) are: The growth from a mine of small tonnage hoisted from shallow depths to one of considerable tonnage derived from all depths down to a mile. . . . Cut-and-fill stoping with square-set filled stoping. . . . The large No. 11 shaft with its numerous levels and complete equipment. . . . Guniting of deep-level crosscuts to prevent scaling. . . . The new No. 12 sub-shaft being sunk from the 3,875-ft. level.

W. F. J. COSSER
Mechanical and Electrical Superintendent

THE pages (485-491) on "McIntyre's Equipment" feature the large No. 11 shaft hoists with the control system. . . . The work of the hoisting ropes. . . . Abundant supply of electricity and the method of purchase on monthly peak demand basis which determines to some extent the use made of electricity and the type of equipment. . . . The electrical cables and equipment in the deep No. 11 shaft. . . . The mill motor controls. . . . The steel sharpening. . . . Daily maintenance of all equipment.

H. G. SKAVLEM
Chief Engineer

MR. SKAVLEM'S contribution on "Engineering and Geology" (pages 451-458) emphasizes the early recognition in the mine's history of the geological location of the orebodies, the strike of the fracturing and the rake of the porphyry. . . . The early, quick action taken in securing mining claims on the strike of the ore. . . . Frank acknowledgment that on every level of this deep mine there still are chances of finding ore as more is learned about the orebodies. . . . The detailed engineering measurements and geological observations made and sampling done, and the detailed records made of these. . . . Close agreement of tonnage and values of ore treated and ore reserve estimates (page 458). . . . The bold exploration programs, based on geological and engineering data, undertaken at depth over a period of years (No. 11 shaft to 4,100 ft.; present No. 12 sub-shaft below the deepest mine levels). . . . Co-ordination of geological predictions, the engineering plans that are based on these, and the provision of mining equipment and facilities to carry out the plans undertaken.

J. J. DENNY
Metallurgist

IN HIS ARTICLE on "McIntyre Metallurgy" (pages 472-484) Mr. Denny stresses the continuous flotation of free gold and sulphides by combinations of new chemicals and use of new machinery. . . . The fine product made by the crushing plant and rolls with three passes, a really remarkable tube-mill feed being made. . . . The individual No. 500 Denver Sub-A flotation machine used at the end of each tube mill. The 75 per cent gold extraction made here with a high ratio of concentration is noteworthy. . . . The small size of the cyanide plant, making possible the small capital expenditure on the mill. . . . Substantial steel-concrete-tile fireproof mill. The lack of belts, the protection for gears and rope drives and the railings everywhere employed all make for safety of men and plant. Low insurance rates are a consequence of this forethought.

A. D. CAMPBELL
Safety Engineer

IN "Safety and Health" (pages 492-496) Mr. Campbell points out the company's regard for the interests of its men as respects fair treatment, wages, insurance, safety and health, and accident compensation. . . . The excellence of living conditions in a normal independent, self-governing Canadian municipality. . . . The safety measures adopted. . . . Reduction in accident rate. . . . The provision of a splendid new change house, modernly equipped. . . . Effective use of sun-lamp treatment for all employees.

Engineering and Geology

The Quest for Gold and the Outcome

H. G. Skavlem

McINTYRE PORCUPINE MINES LIMITED—the name identifies a mining organization, equipped with modern machinery operated under technical direction to extract, at a profit, gold and silver from ores. The success of the enterprise has been due to the coordination of four factors, namely, geological investigation and interpretation; metallurgical research; the use of modern mechanical and electrical devices; and the sociological considerations of a modern community. In this reference to geological investigation and interpretation, and the application of the knowledge thus gained to the problem of mining engineering, we sum up many of the important and decisive facts in the history of McIntyre.

As originally staked, the property comprised one full, 40-acre claim and the land portion of an adjoining claim, a total of 60 acres. The underwater rights of the second claim were then acquired from the Government. The rights of the remaining acreage under Pearl Lake not owned by the Jupiter and Plenaurum Mines were then acquired, and the property comprised 145 acres. By the acquisition of adjacent claims the original acreage was increased to 680 acres. This is in a continuous block and extends for 1⅛ miles along the productive zone of the Pearl Lake area.

During 1911 and 1912, exploration and development of the property consisted of surface trenching, diamond drilling from the surface, and shaft sinking. No. 1 shaft, the original prospect opening, and No. 4 shaft were sunk to explore the area between Pearl Lake and the south boundary. Shafts Nos. 2 and 3 were sunk to develop certain small outcrops uncovered by trenching in the area just north of Pearl Lake.

The Bureau of Mines of the Province of Ontario conducted a geological survey of the Porcupine area in 1911, under the supervision of A. G. Burrows, and a report was issued with maps of the area. This was further supplemented by later field work and a second report with maps published for 1912. Several of the individual companies made extensive surface geological surveys of their properties. That made by the McIntyre covered the ground held by McIntyre Porcupine Mines Limited, the McIntyre Extension Mines, Ltd., the McIntyre Jupiter Mines, Ltd., and the Plenaurum Mines. In 1914 these surveys were combined with those made by adjacent properties and formed a detailed map of the area.

Many interpretations were placed upon the geological data available. Two, in particular, based on work in the Hollinger-McIntyre area, were of primary importance. One was concerned principally with the strike of the veins and mineralized zones. Surface trenching and exploration showed that the strike of the veins varied from north-south at the west to N. 60-70 deg. E. on the east end. Due to this

No. 11 shafthouse. Its height is 174 ft. above the collar

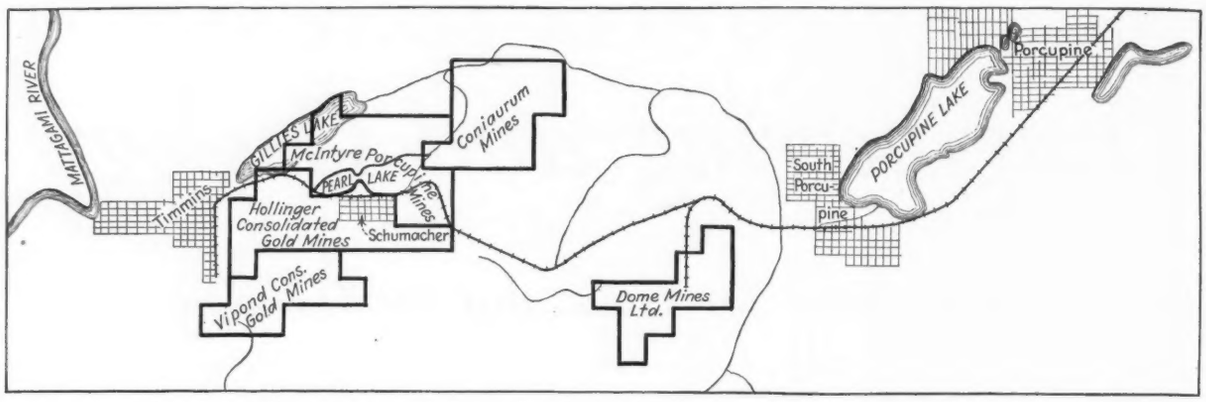


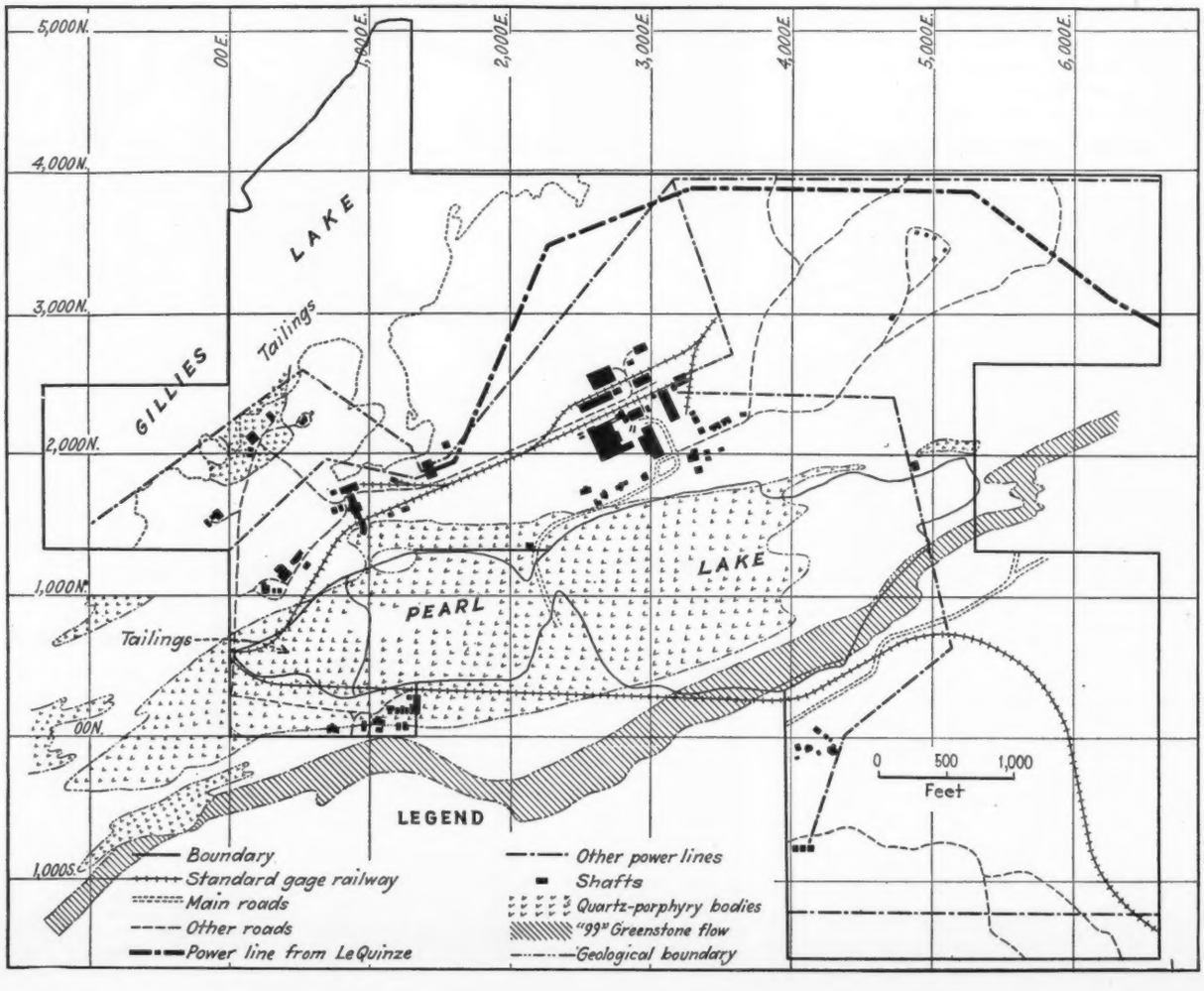
Fig. 1—Map of the Porcupine district showing the five principal properties

great variation in the strike, the idea was formed that the veins and mineralized zones were situated around the periphery of a large basic igneous intrusion, roughly circular in outline and lying southeast of the area being developed, and when the district was fully developed, the mines would be located within a zone of this periphery extending to the east and southeast.

As the development progressed and more was known of the geology, another theory was advanced. As mapped, the porphyry bodies of the area consist of several isolated masses. The largest is lenticular and underlies Pearl Lake, extending to the west of it. It is known as the Pearl Lake porphyry. To the south, west, and southwest are several smaller bodies. These are designated by names.

In the area enclosed and lying between the Pearl Lake, Acme, Millerton, and Miller Lake porphyry masses were found practically all of the important surface outcrops of this district. Underground development around the west end of the Pearl Lake porphyry established the fact of its easterly rake at a vertical angle of 45 deg. The current description of the west end of the Pearl Lake porphyry as it was developed on the Acme mine, now part of the Hollinger property, was that it was like the "prow of a boat." Further development proved this general rake to the east to

Fig. 2—Surface map of the property of McIntyre Porcupine Mines Limited, showing certain geological features



be true for the several smaller masses. Underground development of the extensive surface outcrops indicated the same general relationship between the veins and mineralized zones and the porphyry bodies as at the surface.

This led to the conclusion that the extensive vein development and mineralization were directly associated with and controlled by the location and relative position of the four porphyry masses; namely, the Pearl Lake, Acme, Millerton, and Miller Lake porphyries. The rake of the porphyry would carry this favorable area to the south of the McIntyre property.

On the basis of these theories, the ground held by the McIntyre was not on the strike of the favorable ground.

The development of the area was greatly speeded up by the completion of the Temiskaming and Northern Ontario Railway into the district in 1912. At this time the several properties which now make up the McIntyre were separate mines, and work upon them will be described individually.

Shaft No. 1 was sunk to a depth of 300 ft., and several large veins were developed in the porphyry. Results were poor except close to the west and south boundaries, where the veins entered the lavas. Because of this the stope lengths were short, but some ore was mined.

No. 4 shaft was sunk to 800 ft. to develop the area of lava lying south of Pearl Lake and along the McIntyre south boundary. Porphyry undercut this area, and the amount of favorable ground became less with depth.

Nos. 2, 3, and 5 shafts were sunk in the schistose lavas in the area north of Pearl Lake. Nos. 2 and 3 were sunk to shallow depths, but development was pushed in No. 5. No. 5 vein was developed through it, and below the 400 level showed some excellent ore; above the 400 level it was faulted and erratic. The policy that dictated this work was based on the fact that the orebodies on the south side of Pearl Lake, the chief source of production, occurred dangerously near the south boundary line of McIntyre property, and, the trend and dip of the orebodies being irregular, it was impossible to tell how long they might continue to favor the company as a greater depth was reached.

In consequence, it seemed important to rush development of the orebodies occurring on the north side and lying well within the companies' boundaries.

Early development on the Pearl Lake Gold Mines property consisted of diamond drilling from the surface and several shafts. Only one, now known as No. 6, had had much work done from it. The significance of this work for McIntyre was that neighbors on the north had encountered the company's No. 3 vein (now No. 5) at comparatively shallow depths. McIntyre's report for 1916 stated: "The veins on the north side, while not dangerously near the boundary line, dip towards the property of the Pearl Lake company, and it was of great importance to acquire, if possible, control of the latter. About this time the Pearl Lake company passed into the hands of a liquidator, and eventually all its assets were sold for the benefit of the creditors. In time these properties were vested in the McIntyre Extension Company, and from this new company, McIntyre Porcupine secured the right to use the old Pearl Lake shaft and workings for carrying on its own operations."

The shaft on the Pearl Lake property was continued from 650 ft. to 1,080 ft. A crosscut on the 1,000 level to connect with No. 5 shaft cut No. 5 vein within a few feet of the McIntyre boundary. This was developed above this horizon through No. 5 shaft, and excellent results were obtained. Development of No. 5 vein on the 1,000 level cut a carbonaceous fault, locally called the graphite fault, near the McIntyre western boundary. The possible effect this fault might have on results at depth was viewed with apprehension.

Meanwhile, ground on the north side of the lake and to the east was being developed. Jupiter Gold Mines had sunk two shafts (now Nos. 7 and 8) to the 300 and 475 levels and had opened up some fair oreshoots. Still farther east the Plenaurum had put down two shafts (now Nos. 9 and 10) to the 200 level. A crosscut connecting the two on this level cut several veins, one high-grade oreshoot being developed. The importance of this work was recognized by McIntyre and control of the Jupiter was secured and the McIntyre-Jupiter Company formed.

At this time an extensive program of underground development was undertaken, and much work done on each of the several properties. An option was obtained on the Plenaurum Mines. To explore the Jupiter and Plenaurum ground, No. 7 shaft was continued to the 1,000-ft. level and an exploration drift on this horizon extended well into the Plenaurum property. Results of diamond drilling were encouraging.

A 100-ton sampling mill had been erected at the collar of No. 7. Here all development ore and some production ore from the Jupiter were crushed and transported to the mill. Likewise some production ore from the Plenaurum was handled here. Transportation costs were high, and these operations were discontinued in 1920. As a result of them, the Plenaurum was acquired in 1923.

The only work done on the Platt Veteran claims was trenching and some diamond drilling from the surface. Control was obtained likewise in 1923.

McIntyre was not alone in its policy

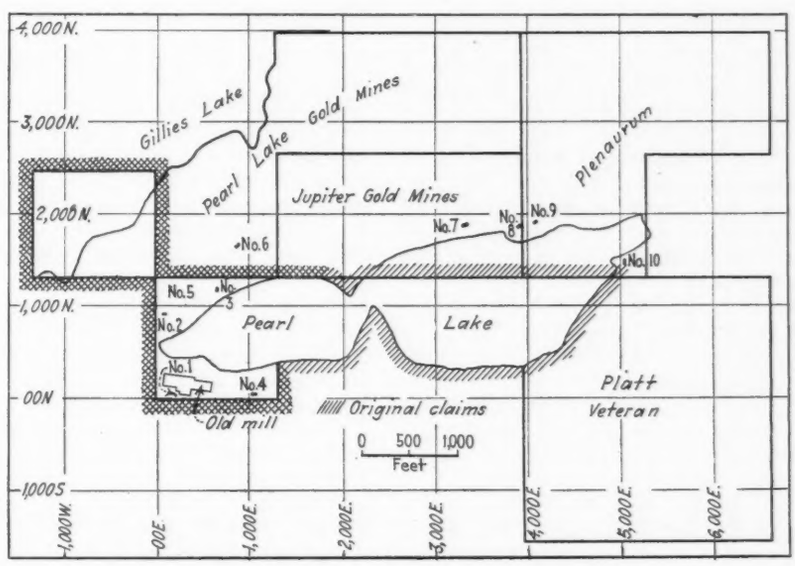


Fig. 3—Original claims of the McIntyre Gold Mines

of expansion. In 1916 the Hollinger Gold Mines, Limited, the Acme Gold Mines, Limited, and the Millerton Gold Mines, Limited, had been amalgamated into the Hollinger Consolidated Gold Mines, Limited. As a result of this centralization there were three operating properties in the Pearl Lake area: the McIntyre, the Hollinger Consolidated, and the Schumacher Gold Mines, Limited. The last was purchased by the Hollinger Consolidated in 1922.

Mining operations up to 1919 had confirmed the following observations regarding Pearl Lake ore deposits: (1) The veins were lenticular, and several lenses might be mined as a continuous mining operation. (2) They were found in a zone of shearing. (3) In plan the various lenses of a vein system were *in echelon*, being usually offset to the left. (4) The lava series was more favorable for ore than the porphyry. (5) Some lava types were more favorable for ore than others. Up to this time but little

data had been correlated with regard to identification of lava flows.

A geological hypothesis was formulated about this time (1919). It is simple, and as development has continued to verify the structure and conditions as projected, our confidence in it has grown. From it evolved the rather daring development program represented by No. 11 shaft, and its achievement. And now another major program is under way, which has for its objective the development of a new mine below the present lowest level of 3,875 ft.

This hypothesis will receive further explanation, but it assumed that the essential seats of ore deposition were related to and controlled by:

1. A zone of shearing
2. Vein fractures or main breaks in the zone of shearing
3. A small angle between the strike of the shearing and that of the fractures
4. Fair continuity of these vein fractures through the zone of shearing regardless of the rock types
5. Basalt and dacite schists more favorable for ore than the porphyry
6. The rake of the porphyry along a line approximately N. 84 E. and at a vertical angle of 45 deg.

On the foregoing assumptions the structural relationship and ground conditions for the area controlled by McIntyre and lying to the northeast of the "very favorable area" were such that extensive ore deposits might be expected.

McIntyre Geology

THE GENERAL STRUCTURE of the Porcupine area[1] is a broad, easterly pitching synclinorium. East and southeast of the Pearl Lake area is an area of Temiskaming sediments which occupy the central trough of the syncline. Lava flows can be traced around their western end. In the Pearl Lake area the alteration has been so great that identification of individual flows is more difficult. South and west of the area a second synclinal structure pitching west is indicated. The areas of Temiskaming sediments in this fold are not so extensive as in the other and the interpretation of the structure is much more difficult.

Save for a few diabase dikes of late Pre-Cambrian age, the rocks of the Porcupine area belong to the lower divisions of the Pre-Cambrian. The oldest are of Keewatin age and consist predominantly of lavas, classified as greenstones (coarse-grained) and basalts (finer grained), together with tuffs and carbonate rocks.

In the Pearl Lake area the various flows have been subdivided in much detail. For purposes of correlation the following classification has been adopted for the lavas:

Vipond Series
Spherulitic Flow
"99" Flow
Central Basalt
Central Dacite
McIntyre Flow
Northern Series

Of these the "99" Flow has a marked characteristic, and the flow relationships are largely identified and correlated through this feature.

There are three ages of intrusive activity with entirely different types of rocks. They are: Diabase dikes, albitite dikes, and quartz-porphyry masses.

The diabase dikes are late Pre-Cambrian, and they cut the lava flows, porphyry masses, and the veins. They have a north-south and a northwest-southeast strike and nearly vertical dips. They are of little importance except as they interfere with operating conditions underground.

The albitite dikes (locally known as "goose egg") are found throughout the central portion of the property north of the porphyry and below the 2,125 level. In general, they parallel the strike of the flows, that is N. 60-70 deg. E., and dip at steep angles. They have marked local variations in strike. They are relatively coarse-grained, with chilled edges, and narrow in width up to 15 ft., but have considerable length, up to 1,000 ft. They cut the lavas and porphyry, but are cut by the veins.

Intrusive into the tilted and folded lavas are several stocks of quartz porphyry, of Algoman age, represented in the Pearl Lake area by several isolated masses. At the surface the main porphyry mass, for this area, underlies Pearl Lake and extends west of it. It is lenticular in shape and about 6,000 ft. long and 1,500 ft. wide in its widest part. Southwest, west, and northwest are several smaller masses. With one exception they rake to the east at an angle of approximately 45 deg., the direction of the rake being about N. 84 deg. E. The exception migrates to the north and joins the main porphyry mass. They are all of the same type.

Development work to date indicates that the proportionate areas of basic schist to porphyry, in the productive area, remain much the same throughout the mine.

The major structures of the older lavas and sediments were dependent on dynamic metamorphism. The Keewatin lavas were folded and partially eroded before the deposition of the Temiskaming sediments in the synclinal trough. Further folding involved both the lavas and the sediments producing the major structure as it is now understood. Minor folds were also developed on the limbs of the major syncline. The zones of schistosity and relief fractures, formed at this time, we believe to be the principal control of later structural developments. The intrusion of the porphyry masses altered the condition of the lavas by creating carbonate and silicified areas in the margins of the porphyry apophysis. The major control features of the primary structure were then further altered by the factors pertaining to the intrusive and the alterations of the country rock by the intrusive.

In the productive area the rocks have been subject to dynamic forces at various times. The complex folding, the fractures now occupied by the veins, the pre- and post-vein faulting, and the broad zone of schistosity are the evidence.

Old No. 5 shaft. No. 6 shaft at left and No. 11 in right background

[1]For a description of the geology of the Porcupine area, see Ontario Bureau of Mines Report, Part 2, Vol. 33, 1924. Also "Geology of the Pearl Lake Area, Porcupine District, Ontario," H. S. Robinson, *Economic Geology*, December, 1923.

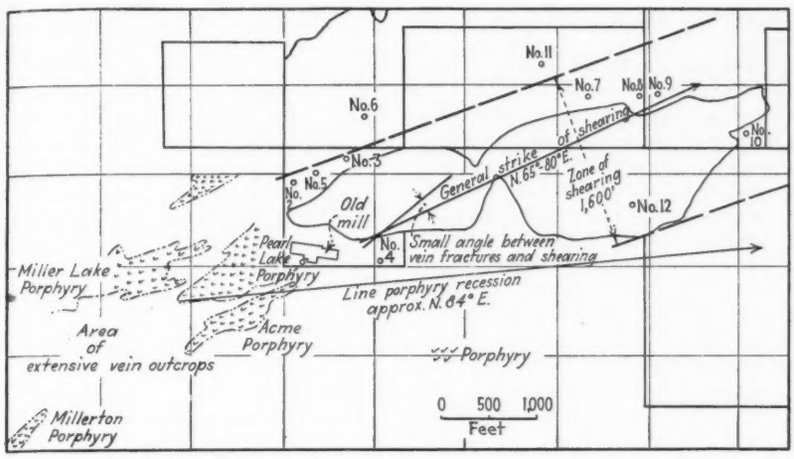
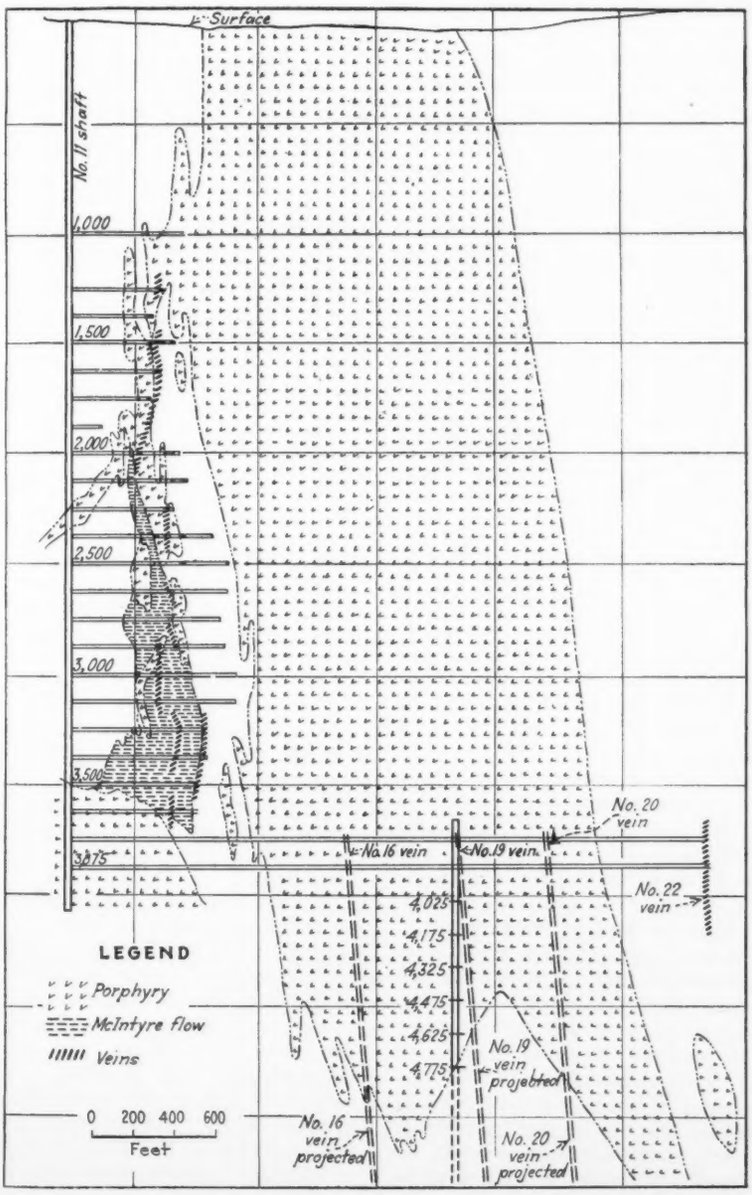


Fig. 4—A generalized sketch showing the structural factors of McIntyre geology

Fig. 5—Section through No. 11 shaft and No. 12 sub-shaft or winze



The zone of schistosity runs northeast-southwest for several miles. Schistosity is not uniform throughout this length, but in it, over a width of 1,600 ft., occur most of the productive veins of the Pearl Lake area. The strike of the schistosity is N. 65-80 deg. E., with a steep dip to the south, except where changed by local folding.

The Pearl Lake porphyry is centrally located with regard to the Pearl Lake area and is within this zone of schistosity.

Local folding in this area is indicated by the fact that to the north of the Pearl Lake porphyry the lava flows face north and are nearly vertical, and south of the Pearl Lake porphyry they face south and dip south. This indicates a minor fold in this area on the north limb of the major Porcupine syncline. The indications are that the Pearl Lake porphyry mass was intrusive along the axial plane of this minor fold.

Further activity occurred after the intrusion, and to the regional stresses were added those set up by the cooling porphyry masses and by the alteration effect of the porphyries on the intruded lavas. During this period the physical character of the lava flows and their attitude had a marked influence on the degree of fracturing of the schistose and non-schistose areas, both for lavas and for porphyry.

Fracturing occurs on several planes throughout the zone of schistosity. They are identified on the McIntyre by our several vein systems. These fractures cut across the strike of the schistosity at a small angle to the left. Development work on veins in the lavas, and diamond-drill results in the intervening porphyry, indicate the fair continuity of these major breaks throughout the length of the McIntyre property, regardless of type of rock. Where such fractures are in the zone of schistosity and in the lavas in proximity to the porphyry, or in the lava-porphyry contacts, the structural relationship is favorable. In such areas the principal ore occurrences are found.

Development has proven vein material to occur in these major breaks in all the rock types or their derivations in the area. The major part of all ore developed to date, however, has been in lava schists of a friable character and in or relatively close to a porphyry contact. Ore is found in narrow porphyry masses and passing into the main porphyry masses if the vein is strong and the tenor relatively high in the adjacent lava schists. Although no great amount of development has been done in the large porphyry masses, the veins, where cut even in the main porphyry mass by exploration crosscuts, have in some cases shown gold, but were not ore.

At least six of these major breaks occur within the zone of schistosity in the McIntyre property.

As development has proven the most favorable ground for ore to be along

these breaks in the lava schists and lava-porphyry contact schists, it follows that any areas likely to have all or most of these favorable characteristics must be explored. For the McIntyre, there are two such general areas:

(a) The lava schists south and west of the main Pearl Lake porphyry mass, or on its underlay side

(b) Lava schists north and east of the main Pearl Lake porphyry mass, or in general above this porphyry

For area (a) the breaks or veins which are in porphyry in the upper horizons of the McIntyre will pass into lava schist at lower horizons, and for each successive break to the south across the porphyry this point of emergence will be farther to the east and at a greater depth. For area (b) these conditions are present from the surface to the deepest level of the McIntyre.

McIntyre's Policy

The development policy of the McIntyre has been to explore these areas.

Prior to 1918, development was primarily to develop the surface showings of the several separate companies, and work was continued along the same lines even after the various properties were consolidated into one unit. Shafts Nos. 1 and 4 and 2, 3, 5, and 6 were in area (a), and shafts Nos. 7, 8, 9, and 10, in area (b).

Since 1918, the mine has been developed with the idea of exploring the favorable areas as outlined by the geological hypothesis explained previously.

No. 5 shaft developed area (a) to the 2,375-ft. level. No. 6 shaft developed area (a) to the 2,875-ft. level, and connections were made through to No. 5 shaft on the 400, 600, 1,000, 1,375, and each succeeding level to the 2,375-ft. level.

Nos. 7, 8, 9, and 10 shafts develop area (b) and No. 11 shaft provides facilities for continuing this development below the 1,000-ft. level to the 3,875-ft. level. Connections are made with No. 7 shaft on the 1,000-ft. level and with No. 6 shaft on the 1,500-ft. and each successive level to the 2,875 level. Below the latter level No. 11 shaft provides the facilities for the development of both areas (a) and (b).

Development footage was as follows to March 31, 1933: Shafts, 12,859 ft.; crosscuts, 97,038 ft.; drifts, 199,116 ft.; and diamond drilling, 353,714 ft., or a total lateral development of approximately 60 miles.

Much favorable ground remains unexplored, for which the present shafts and equipment will provide ample facilities. Recent work in old parts of the mine indicates that much salvage will be obtained on remining these areas both because of better facilities to handle the ore and new ideas as to the orebodies.

Development crosscuts southeast of No. 11 shaft on the 3,750- and 3,875-ft. levels through the Pearl Lake porphyry and into the lava schists on its south side have proven several breaks in the main porphyry body and a parallel series of breaks to the south of the porphyry in the lava schists. At these horizons development of the break farthest south has proven ore for a length of several hundred feet. Diamond drilling from the 3,875-ft. level has proven ore in this break to the 4,400-ft. horizon.

The new No. 12 shaft, an internal shaft, is now being put down. Its ultimate depth will be 3,000 ft. and it will provide facilities for developing and mining to a depth of 6,900 ft. below the surface.

There are two types of ore: (A) Quartz veins with schist inclusions. (B) Irregular replacement veins. An orebody may be of both types.

The orebodies being in fractures, which may be found in all the rock types of the area, wall rocks are in consequence any of the rock types or their derivatives. Of these it may be said:

1. Basalts may show any type of alteration to any degree.
2. Greenstones and dacites may show any type of alteration to a much less marked degree than the basalts.
3. Porphyries are not chloritized.

The majority of orebodies are found in types 1 and 2.

Ore in veins which pass from the porphyry into the lavas usually grades from Type (A) in the porphyry and contact zone to Type (B) in the lava schists away from the porphyry. There appears to be a zone limit to the occurrence of orebodies and the porphyry masses. Most deposits are found within 1,200-1,500 ft. of porphyry.

The type of orebody and the type of wall rock change within short distances. This has an important bearing on the method of mining to be used. Basalt and porphyry contact areas are usually highly carbonated and sericitized. In areas characterized by the latter alteration the wall rocks tend to slab off, and dilution is serious; close wall support is necessary.

In ore of Type (B), or the replacement type, the mineralization tends to follow joint planes and fractures into the walls. For clean mining, close wall support is necessary. In local areas faulting has in some cases necessitated back support as well as wall support.

Methods of mining in use are shrinkage, cut-and-fill, and square-set and fill. At present about 60 per cent is by cut-and-fill. This percentage will increase as wall stresses increase because of greater depth of mining.

The lenses of ore as mined are usually *in echelon;* that is, in plan, successive lenses step to the left. As a result, no great length of ground is opened up in a single plane. Some lenses have been mined for a length of 1,200 ft., but the average is less. Because of this structural arrangement, the tendency for accumulation of pressure or stress is less than when mining is conducted along a single fracture or plane for considerable distances. The character of the wall rocks is also such that they yield to stresses gradually; no cases of explosive violence due to relief of rock stresses are known. As greater depth is attained, the difficulties presented by wall-rock stresses will increase, but to judge from past experience, they will present no serious hindrance to mining operations.

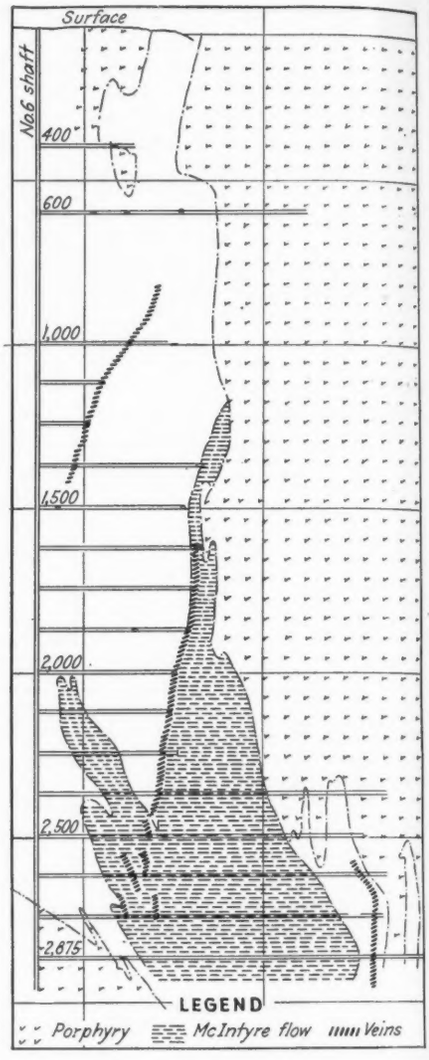


Fig. 6—Section through No. 6 shaft

The temperature gradient of the rocks of the Porcupine area has been thoroughly covered by R. H. Cleland in a paper presented at the annual meeting of the Canadian Institute of Mining and Metallurgy, held in Toronto last April. The following figures are from his observations. The gradient, one may note, is 1 deg. F. for every 223 ft. of vertical depth. Surface rock temperature is 40.5 deg. F.

Average humidity was found to be

about 98 per cent, the dry-bulb temperature ranging about 2 deg. F. above the rock temperature and the wet bulb from 0.5 to 1.5 deg. F. above.

Both pre- and post-mineral faults occur in the area. The former are important because of their control, in instances, of mineralizing solutions. The latter are practically all pressure faults, and their displacement is not great but is enough in some areas to increase the difficulties of mining.

Of most importance is a zone of faulting which strikes about N. 70 deg. E. and dips 60-70 deg. southeast. In it is a parallel series of south-dipping faults, and these are again intersected by a minor series of north-dipping faults. At the intersection of these faults there is some displacement and a carbonaceous development, the carbon in some areas running as high as 7 per cent.

This fault zone is of particular importance in that it intersects three of the principal vein systems of the McIntyre. It was mentioned previously as a feature of critical importance when it was cut in No. 5 vein system at the west end of the mine on the 1,000-ft. level. In the zone of intersection the original vein material has been broken up and displaced so that there is much dilution with any method of mining. Also, this zone is characterized by an amorphous carbon development, which, in association with other than cubical sulphide combinations of iron or copper, has a marked precipitating effect on gold cyanide solutions. Should a small tonnage of this material be sent to the mill, even on a 2,000-ton per day basis, its presence is quickly noted in the mill results. Research has devised a method of treating this ore with excellent recovery.

The principal vein fractures when intersected by this zone appear to break up into a series of small fractures. Extensive replacement ore deposits are local to this condition.

The McIntyre mine is comparatively dry. A general watercourse occurs which has been tapped by some exploration crosscuts and diamond-drill holes. These have been sealed, and, as the location of the course is now fairly well established, it has given little difficulty, with care in directing development work.

Level intervals are 100 ft. to the 1,000-ft. level, thence to the 3,875-ft. level 125 ft. Below the 3,875 level in No. 12 shaft the interval will be 150 ft., as all stoping below this level will be cut-and-fill and the need of intensive exploration limited because of a better understanding of geological features.

Because the orebodies are successive lenses in a series of parallel fractures, and usually step to the left in plan (as already said), and also have the same attitude in vertical section, development is largely concerned with picking up the next succeeding lens in a parallel break. In production, when a lens terminates between levels, the problem is to find its bottom. In either case the diamond drill has been an important aid and has been used extensively for this purpose.

Exploration is controlled largely by geological considerations. Certain areas are known to be favorably located; these are developed by crosscuts and drifts without preliminary drilling. Doubtful areas are first explored by diamond drilling before a major development is started. Certain areas are drilled with the idea that negative results will be conclusive. No policy of development by crosscuts at stated intervals of say 300 ft. has been adopted by the McIntyre, but rather a policy of line drives to certain areas on various levels has been the program.

When such an area is developed by crosscutting the favorable ground, the custom is to drift on the strongest break exposed in the crosscut. This is opened up to its limits. Should there be a break or an indication of a feeder for the next lens, it is followed. If not, a drift is usually advanced 50 ft. or more on the line of the general strike of the vein, and diamond-drill holes are then put out to explore the possibilities of the parallel breaks. Development then proceeds according to results obtained.

In conclusion it may be said that the several development campaigns inaugurated by the McIntyre have called for large capital expenditures. Throughout, such campaigns have been aggressively pursued through the whole-hearted support of the management and directors.

Sampling and Estimation

PORCUPINE ORES can usually be identified and followed in ordinary mining operations. But in some instances good-looking vein material is not ore and in others very unpromising-looking material is ore. Extensive sampling is therefore necessary.

All development faces are sampled each round. Half of these samples are taken with the muck from the previous round still in the face, so that any one of them can represent only the top third or half of the face.

No sample is longer than 5 ft., and usually the face sample is sectionalized to indicate the distribution of values. The sampler soon becomes expert in identifying the pay sections. The first sample is always taken on the north side of the face. Notations are made by the sampler at the face as to sample length, percentage of quartz, degree of mineralization and schistosity, and visible gold.

All crosscuts are sampled, and in 5-ft. intervals. These samples are taken on the east rib and check samples on the west rib. In stoping, an effort is made to sample each breast as broken.

Test holes are drilled to direct slashing operations in stopes. They are usually marked up by the bosses.

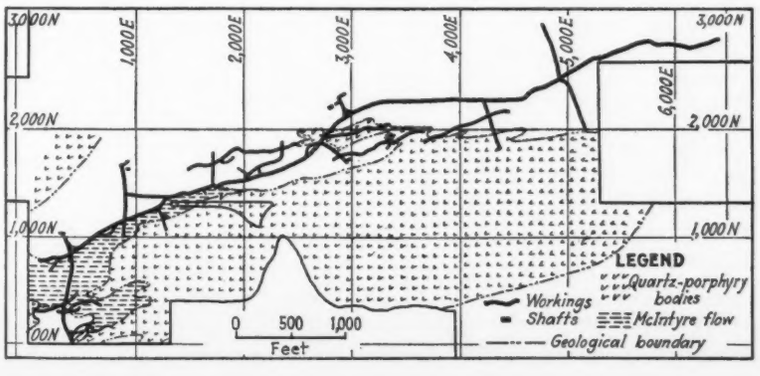
Where considerable slashing has been done, each slash is sampled, but to facilitate assay calculations a back sample is taken across the full width of the vein at 10-ft. intervals after the vein has been slashed to full width. It is customary to check-sample the back of drifts, when in ore, at 10-ft. intervals.

Box or muck samples are taken from the individual stopes from which ore is being drawn and from each development drift face. The check between these samples and channel sample results is comparatively close.

Many of the present mine operating officials have risen through the ranks of the sampling organization. The close association with working places throughout the mine is of great value in directing day-to-day development.

Estimation of Ore Reserves—The Assistant Mine Engineer is in immediate

Fig. 7—Sketch showing workings on 1,875-ft. level, in relation to the quartz porphyry and the McIntyre Flow



touch with all actual development and production faces. This knowledge is of importance in estimating reserves and control of underground operations.

For ore-reserve estimates, the usual assay calculations for width and length are made. Each block is evaluated, using these calculations with due consideration given to geological evidence.

As a stope is mined, back samples supply a new basis for assay calculations for the block of ground remaining. Stope and muck samples are checked against the reserve estimate of the ground broken.

The weight given to high assays depends upon their frequency and proximity to each other. Lone, erratic high assays are usually arbitrarily cut to $50.

Over a period of fifteen years, tonnage treated being 6,140,523, the yield of value recovered as checked against ore-reserve estimate has been about 1 per cent low.

In mining an attempt is made to budget the ore to the mill from each vein in proportion to its percentage in the ore-reserve estimate. Since the inception of mining, stopes have been found to yield 13 per cent excess by tonnage and 17 per cent excess by value over survey and office calculations.

After an allowance of 10 per cent for line drives has been made, 46 per cent of drifting is in ore.

Diamond Drilling—The first diamond drilling at the McIntyre was in 1911. This was from the surface and was by contract. The first underground drilling was done in 1914. All drilling since that time has been on company account except for a few holes drilled on a contract basis during 1917.

Operations may be classed as exploratory and operating. Exploratory holes are usually long, up to 2,700 ft. The great majority of holes to direct operations are less than 250 ft.

The average operating cost, without supervision, totals about $1.16 per foot.

Geological Data—Correlation of McIntyre geological data up to 1930 was the work of several mine geologists. In that year a full-time geologist and one assistant initiated a complete check-up on geological data. The plan was to commence with the bottom levels and work up through the mine. In this way previous conclusions would be confirmed or discarded by developing the picture of McIntyre geology through another avenue of approach. Greater attention was to be paid to detail and the classification of rocks throughout would be by the same individuals.

All available workings were mapped, the walls were washed down for detail mapping, and all drill cores were relogged. Mine maps on a 30 scale were copied on sheets 7x11 in., and these were used underground. The geology and all notes were put on these sheets, and they constitute the only field notes. The data from them were plotted on 200 scale maps on which were the drill-hole logs. Sections to the same scale, and at 150-ft. intervals, were made covering the whole property. By this means reasonably accurate interpolations have been made for completing the maps. During this work much microscopic work was done in studying the vein mineralogy, and as an aid in identifying obscure rock types. A model of the mine was made from the 200-scale sections. This is of the vertical section type, and plate glass sheets are used.

This work has produced a complete geological picture of the mine, and as a result new ideas have come forward—some empirically and others through theoretical considerations—which have been used in prospecting new areas of the mine.

After the completion of the generalized geological work, the routine was changed to keep the data up to date. As organized at present this entails mapping development headings and stopes.

In particularly intricate parts of the mine small sectional glass models are made. They are on 10x10-in. plates and to a 50-ft. scale. A cubic block 500 ft. to a side is shown on one model. When not in use the plates are filed. These models are of assistance in development and production operations, and also in laying out diamond-drill holes.

In search of another McIntyre
H. S. Robinson, geologist

Field Exploration

PROSPECTING has played a most important part in the rapid development of the mining camps of northern Ontario, Quebec, and Manitoba. Each year individuals, syndicates, and exploration and operating companies enter the field and explore the likely areas of favorable formation.

For the last ten years the McIntyre has maintained a separate department for outside exploration. This has been in charge of H. S. Robinson. The work of the department may be classified as: (1) Prospecting and staking. (2) Exploration and development of prospects. (3) Examination of properties offered for sale.

1. Each year, beginning with the spring break-up, one or more parties are sent out, with equipment and funds for six months, to prospect a selected area. A party consists of two men, one an experienced prospector and the other his assistant. Before leaving, the prospector is supplied with all the available maps. Details of the proposed trip are discussed with the engineer in charge of exploration. The party is equipped with licenses and proxies to enable them to stake 18 full 40-acre claims. They usually stay in the field until Nov. 1, reporting by letter when possible.

2. Exploration of claims staked by the company's prospectors usually starts shortly after recording the claims, as required by the regulations of the Ontario Department of Mines. The character of the discovery determines the kind of work. Usually, a party of five to fifteen men in charge of a young engineer moves in, puts up a temporary camp, and proceeds to make a geological map of the ground. At the same time test pits are put down on the showings and rock trenches shot out along the strike to determine as far as possible its extent. If the preliminary sampling indicates an orebody, it may be further explored by diamond drilling or underground work.

3. Many properties are offered for sale each year. Those that appear worth investigation are examined by the engineer in charge. A full, written report is made, even if results are negative. Recommendations are made to the general manager with the report and appropriate requisitions for proposed work.

Mines are difficult to find and the element of chance still seems to be the predominant factor.

The story of McIntyre—the lure of gold—is truly one of enthralling interest; the satisfaction of pioneering both in a new country and in new methods is ours. For the future there is still the hope of adequate reward for similar endeavor.

Development and Mining

Seven Thousand Feet the Objective

D. E. Keeley

Trolley locomotive and Granby cars dumping on the 2,875-ft. level

UP TO THE TIME that the 1,000-ft. level crosscut from No. 6 shaft was connected with the workings of No. 5 shaft, as told elsewhere (page 453), mining practice had been to drive small crosscuts and drifts (the first Leyner-type machines were replacing the piston machines on this work). Shrinkage stoping had been done with hand-rotated, dry stopers. The ore had been handled in 16-cu.ft. cars, and ore and waste hoisted in them in cages to the surface.

In 1917, the daily mill tonnage was increased to 500. Underground methods were accordingly altered to suit larger-scale operations. All mine openings were made larger, grades for haulage improved, larger cars used and older underground drilling equipment was replaced with more efficient and improved machines as fast as they were perfected. The rapid development of increased efficiency through improvements in machine drills and steel had much to do with speeding up mine development.

The first 1½-ton automatic transportation storage-battery underground locomotive was purchased in 1917, together with faster roller-bearing cars. Skips replaced cages for handling ore and waste.

An ore pass was completed from the 600 to the 1,000 level in the No. 5 vein section. All ore was passed to the 1,000 level, loaded in cars, and trammed to the Pearl Lake or No. 6 shaft, and dumped over a sledging grizzly into an ore pocket. From this it was loaded directly into 2-ton skips and hoisted.

A new crusher house was erected at this No. 6 shaft and an aërial tram built from there across the lake to the mill. Nos. 5 and 6 shafts were then deepened by 500-ft. steps, the former to 2,405 ft. and the latter to 3,024 ft., this being for some time the deepest in Ontario.

The general plan of mine development below the 1,000-ft. level was to establish levels 125 ft. apart with a main level every 500 ft., with a loading pocket for ore and waste at this point at No. 6 to accommodate a 500-ft. vertical section of the mine. Branch ore-pass raises were put up to the intermediate levels, and at each main level above the loading pocket were placed grizzlies with 14-in. openings, where all ore was broken by hand.

Main haulage levels were equipped with 30-lb. rails and 18-in. gage track, still uniform throughout the mine. Whitcomb 2½-ton storage-battery locomotives were put in service on the main haulage levels. To supplement main haulage on intermediate levels, 1½-ton storage-battery motors, folding to a length of 4 ft. 4 in. for cage transportation, were installed.

In 1924, owing to the eastward trend of the orebodies with depth, decision was made to sink a new shaft to facilitate mine development. This was located 2,000 ft. east of the then main hoisting shaft, No. 6. Station levels were required at every 125 ft. of depth in the new shaft below the 1,000 ft. Considering this, and various factors of rock and ore conditions, tonnage, power, time, and safety, a depth of 4,100 ft. was decided on as the maximum economic operating depth for the shaft. Deeper development would have to be done from internal or sub-shafts.

No. 11 shaft was accordingly started Oct. 14, 1924, with an excavation size of 17x24 ft. at right angles to the schistosity, and was completed to a depth of 4,133 ft. on March 5, 1927. McIntyre No. 6 shaft, as already mentioned, had previously been the deepest in Ontario.

The work of constructing, equipping, and placing this deep No. 11 shaft in operation as a unit was a big task. It involved the sinking and timbering of 4,133 ft. of large section, vertical shaft; the cutting of large stations at 125-ft. intervals below the 1,000-ft. level; the running of exploration and connecting crosscuts from the stations; the equipping of shaft and levels with service lines for compressed air, water, and electricity; the raising of ore passes with their branch raises and control chutes from the 4,100-ft. level ore pocket to the 300-ft. level; the placing of a large, coarse-crushing plant below the deepest working level; the erection of the 175-ft. steel-frame shafthouse with its ore bins; the placing of modern high-speed hoists on surface, the one to hoist ore from the deep ore pocket to surface and the other to hoist all men and materials from 25 mine levels.

In the course of sinking this shaft in 28 months, the temporary headframe and surface equipment were erected, 400 rounds removed, timbering to the bottom was completed, including 23 sets of bearer timbers 10x24 in. by 20 ft.; nineteen stations were cut and timbered, and 600 ft. of crosscutting was done for a total of 4,133 ft. of shaft. An average

of 16.3 men per shift were employed below the collar during sinking operations. A summary of the work done[1] is given in Table I.

Twelve to fourteen machines were used in the shaft. Pieces of steel required averaged 113.5 per round of 771 ft. of drilling, in double and triple V-cut rounds. Ventilation was provided by a reversible exhaust-blower 10,000-cu.ft. rotary fan, driven by a 75-hp. motor. A 20-in. ventilation pipe was run from the fan down the service compartment. Muck was hoisted in 1.6-ton tapered-side buckets at 1,850 ft. per minute. Equipment features were the air and water headers and the steel box for handling steel.

All shaft sets used were made of Douglas fir, surfaced to size, all carefully framed by machinery. The framing shed was near the shaft collar.

A guaranteed wage of $5.80 per shift was paid, plus a monthly bonus calculated on shaft advance at approximately 25c. per foot. The average wage to all men working below the collar of shaft was $7.98.

Level Interval 125 Ft.

The levels of No. 11 shaft are spaced 125 ft. apart, adjustments being made to connect at grade with levels from older shafts. The 23 stations are cut the full width of the shaft for a distance of 12 ft., and station crosscuts 9 ft. wide are driven beyond. Stations and adjacent crosscuts are gunited and whitewashed. Development from this point depends on locations of oreshoots at that level.

The ore-pass system extends continuously at a pitch of 55 deg., direction reversed, at each level from the loading pocket at the 4,050 level to the 300-ft. level. A 25-ft. branch raise connects each mine level with the main ore pass. These branches hold about 100 tons of ore each, and, being driven from the back of the main pass, the ore in them goes down first when the pass is drawn. The main pass can thus be kept filled with a large tonnage of ore and still leave available dumping space on each level. Finger control gates are placed every 500 ft. vertically in the pass. The system holds 12,000 tons of ore.

Just below the 3,875-ft. level and in the ore-pass system is the 42x48-in. crusher, described elsewhere (page 474). Below it a storage pocket extends to a loading pocket.

Since 1930, all ore from the west end of the property has been transferred to No. 11 shaft and, with the remainder of the ore, has been hoisted to surface in 6-ton balanced skips at 3,000 ft. per minute. The hoisting equipment is described elsewhere.

No. 11 shaft has made it possible to handle a large tonnage and to continue development at depth. Work extends for the full length of the property, and production has come from all levels. The old sections of the mine are now supplying, and will for some time supply, much of the mill tonnage. No level may be said to be completely developed, even after 22 years of active operation.

Mining methods during the life of the mine have changed greatly. In the early years all stoping was by shrinkage, which gave low costs so long as the stopes could be kept filled with broken ore. By 1924, increased depths, with their increasing rock pressures, with consequent dilution of the ore by barren wall rock, made necessary a change from shrinkage stoping to more precise methods of extraction.

After careful study, horizontal cut-and-fill was adopted as the standard method of mining those orebodies which could no longer be handled efficiently by shrinkage. It was found necessary to supplement cut-and-fill methods by square-setting and filling certain stopes where extreme widths and condition of roof and walls demanded additional support. Inasmuch as the haulageways usually follow the veins, some standard way of mining the floor pillars and at the same time keeping open the level had to be worked out. It was found most satisfactory to complete the mining of such floor pillars by square-setting and filling the last 16 ft. of roof. During the last three months, a modification or a combination of square-set and cut-and-fill has taken the place of the standard sets with equal safety and added economy.

Evolution of mining methods has been a gradual growth, and great credit must be given to the individual miner for his willingness and intelligence. He has been trained to, and has himself reached, a high degree of proficiency.

Great credit is due, also, to the rock-drill manufacturer. When the present high-power type of machine is contrasted with either the first types of similar machines, or with the 4¼-in. piston slugger of earlier years, one must recognize that tremendous strides have been made, and the mine operator must acknowledge his debt.

Another important help to present-day mine operation is the greater use of the diamond drill. Small portable machines are used extensively in solving detailed structural problems, and much ore that otherwise might be left is found.

Mining Methods — All development has been done from vertical shafts. The most important are listed in Table II.

Level intervals to 1,000-ft. depth are 100 ft. vertically; from 1,000 to 3,875 level, 125 ft.; and below the 3,875 ft. level, 150 ft.

The standard opening for drifts and crosscuts is 8x8 ft. A standard round consists of from 20-24 holes using a diamond or pyramid cut. Cut holes are 8 ft. deep. A round averages over 6.5 ft.

Grades are driven at 0.5 per cent in favor of the load. The gage is 18 in., and 30-lb. rail is used, with standard railroad type of frogs and switches.

Leyner-type rock drills of several makes with 3-in. and 3½-in. pistons are used in both development and production, and self-rotating wet stopers in raises and box holes. A driller and helper set up and drill a face and blast in 8 hr., air and water are then blown for 4 hr., and two muckers clean out the round in the following shift. Two shifts are worked—from 7 o'clock to 3, night and day. At times when speed has been wanted in a single heading, four 6-hr. shifts, using two machines and three men on the drilling shift and a similar crew on the mucking shift, have given an average progress of 13 ft. per day. Where a slusher outfit is used, the same daily advance is made, with four men per shift and two shifts per day. Two machines are used, each crew mucking out, setting up, drilling, and blasting in 8 hr. Advance of 18-20 ft. per day can reasonably be anticipated in a single heading, with three 8-hr. shifts and a mechanical loader.

Orebodies at the McIntyre cannot be mined from a blueprint. Although certain methods are used, each stope is a problem in itself, and at times a change from one type of extraction to another is necessary as conditions warrant. Frequently, all mining in a particular stope must be halted until parallel and blind lenses of ore in the walls are explored. Minor faulting and folding are common, more so on the upper levels, and complicate the problem of mining. In certain areas, the hanging wall is badly broken and comes away in large blocks, causing difficulty in the chutes, and shutting off that portion of the stope.

The three methods of mining used have been mentioned. Often a combination of all three is necessary in working one stope. The following details respecting stoping are by S. A. Wookey, Assistant Mine Superintendent.

Shrinkage Stoping—Before a stope is begun, all information regarding the

Table I—No. 11 Shaft Sinking Data

160,000 tons of rock removed, hand shoveling (5 tons per man-shift)	
40,000 tons of water, hand baling	
304,000 ft. of drilling, 10 ft. per man-shift	
27,000 holes drilled	
240,000 lb. of 40 and 50 per cent gelatin explosive used (1.5 lb. per ton of rock)	
2,256,000 ft. of timber placed	
300 cords of cedar lagging placed	

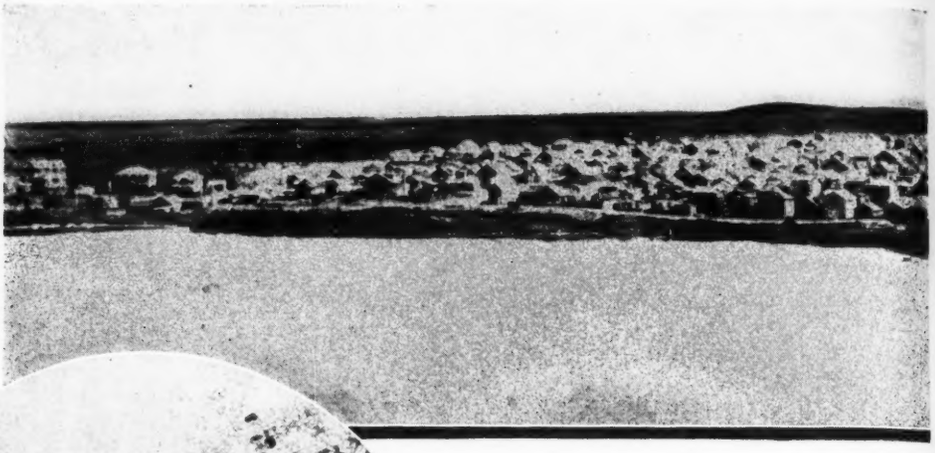
Table II—Shafts, McIntyre Porcupine Mines

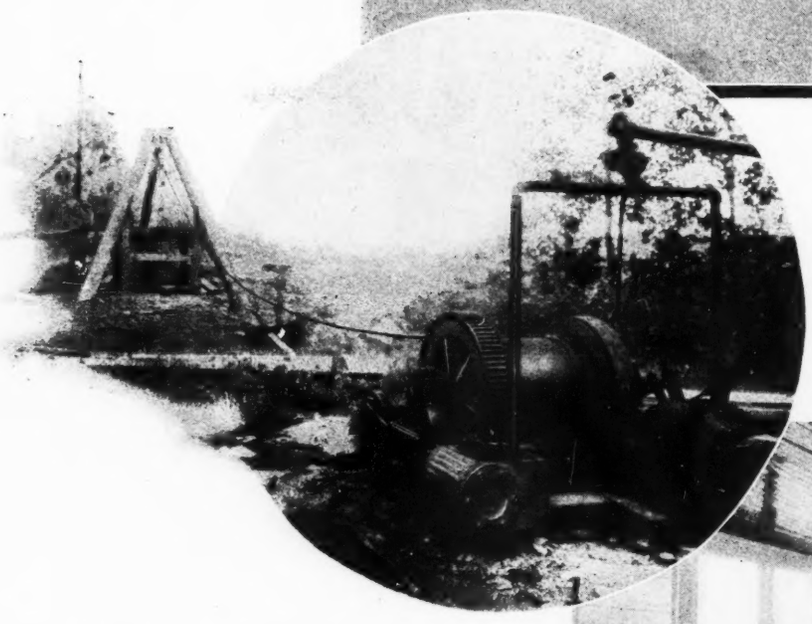
No.	Location	Depth, Feet
4	Southwest corner of property	820
5	Northwest corner of property	2,388
6	825 ft. northeast of No. 5 shaft	3,015
7	650 ft. southeast of No. 11 shaft	1,000
11	2,000 ft. east of No. 6 shaft	4,131
12 (Winze)	2,000 ft. south of No. 11 shaft....Sinking below 3,875 ft. level	

[1] H. A. Kee (C. I. of M. & M., 1928).

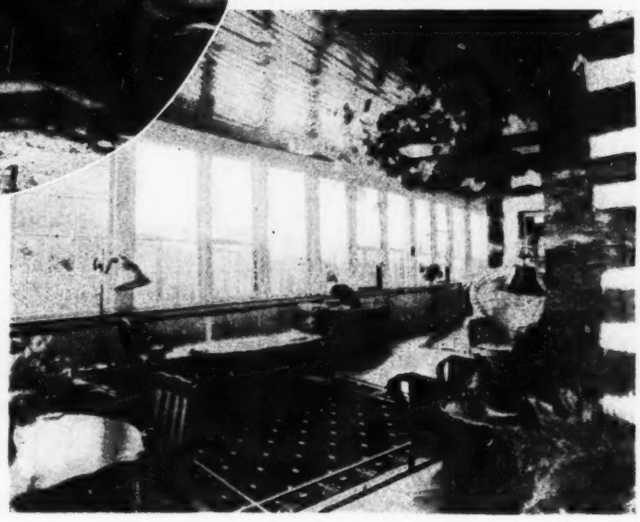
Sunset at No. 11 shaft. Here the dark bulk of the lofty shaft-house and its idler stand dominate the landscape.

A glimpse of the underground world below.





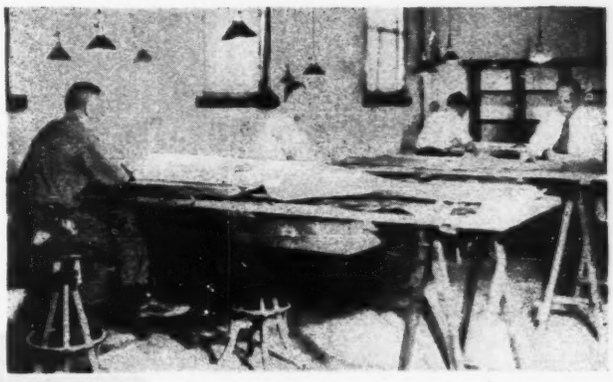
An attractive retreat. The sun porch of the general manager's residence, in the woods near the plant.



Not only at depth does McIntyre's future lie, but from the surface down. Here after twenty years of underground operation mining is beginning on an outcrop on the property. Every level is thought to offer similar possibilities.

A distant view of McIntyre's centralized surface plant at No. 11 shaft. The nature of the surrounding country—wooded and of a low relief resulting from glaciation—is here apparent.

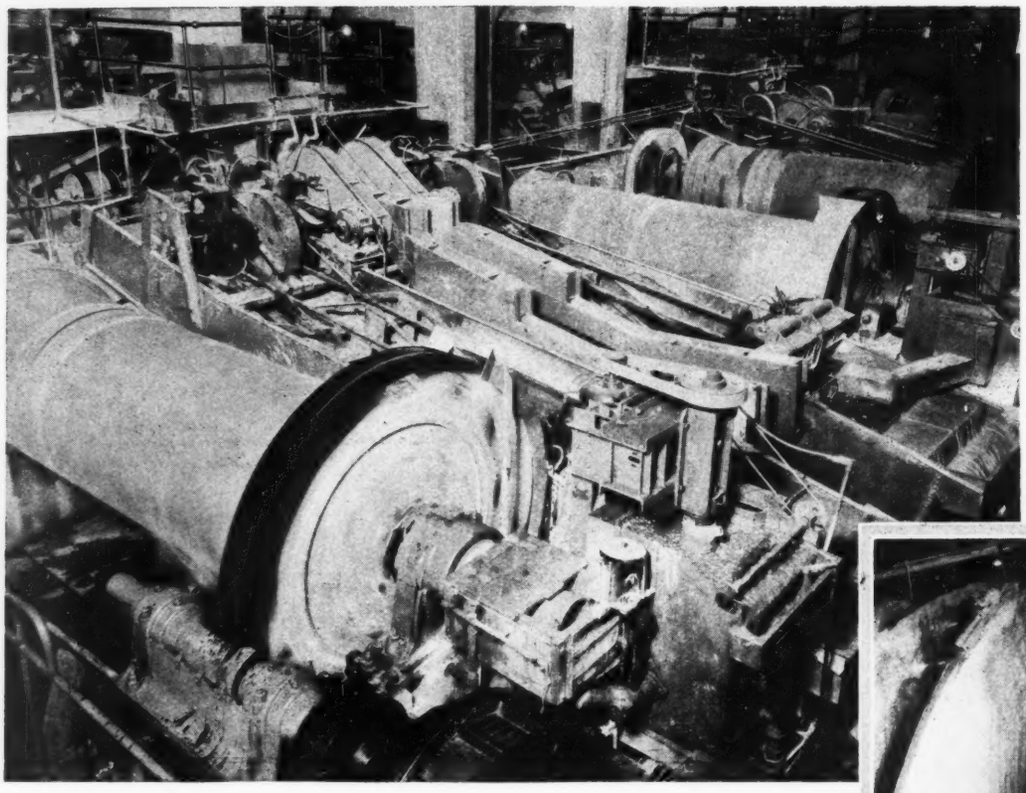
Across Pearl Lake from No. 11 shaft lies Schumacher. Not a company town but, instead, a normal self-governing Canadian municipality and a community of comfortable homes.



In the administration building—a corner of the engineering drafting room. The same structure houses the offices and the modern change room.

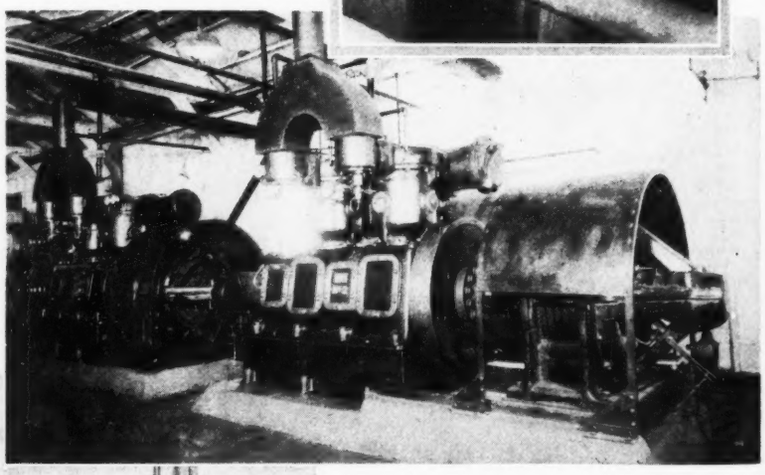
Some of the commodious staff houses near the plant. The setting is one of many shades of green.

St. Mary's Hospital, in Timmins, a mile from No. 11 shaft. A community facility which the McIntyre helps support.

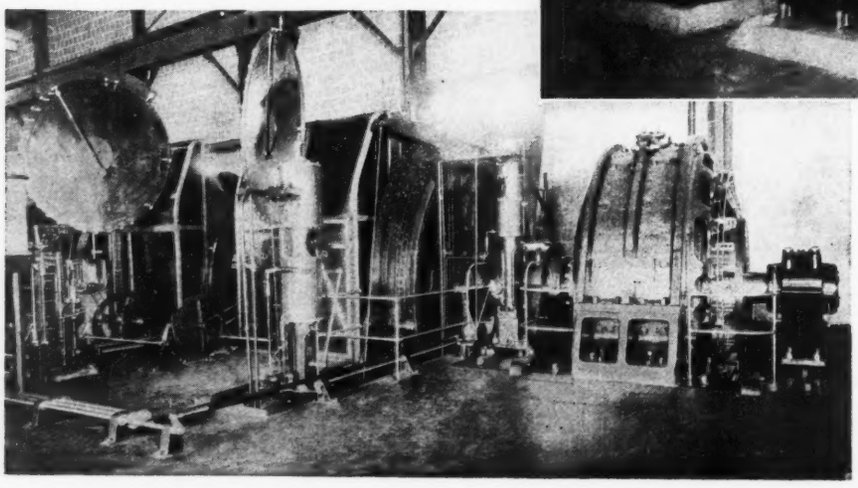


In McIntyre's concentrator — Tube mills grinding ore in closed circuit with adjacent classifiers. The lone flotation cell in the right foreground is an outstanding feature of McIntyre metallurgy.

Filter leaves close-up, in the filtration section of the mill.



The ore hoist at No. 11 shaft. This winding engine is equipped for every contingency with up-to-date safety devices.



Two of the units in the compressor building at No. 6 shaft, whence air for all underground work is supplied.

(Continued from page 460)

probable dip, condition of walls, width, grade, and related information is summarized and the decision made as to the method to be employed. Sometimes a stope is mined by shrinkage 50 to 60 ft. above the rail, cleaned down, the chutes raised and fill run in, and the remainder broken by cut-and-fill. Normal procedure in starting any stope is to take down the back to a height of 18 ft. above the rail over the full width of the vein, or to drive box holes spaced at 25-ft. centers, leaving a pillar 12 ft. thick and joining the box holes by sub-drifts at this elevation and carrying on with regular stoping above.

Where the backs are taken down, 12 to 18-in. fir or round peeled spruce of

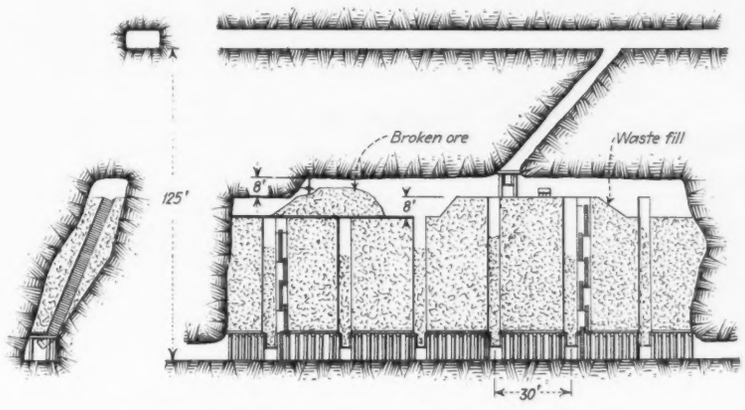


Fig. 1—Longitudinal vertical section (sketch) through a standard cut-and-fill stope

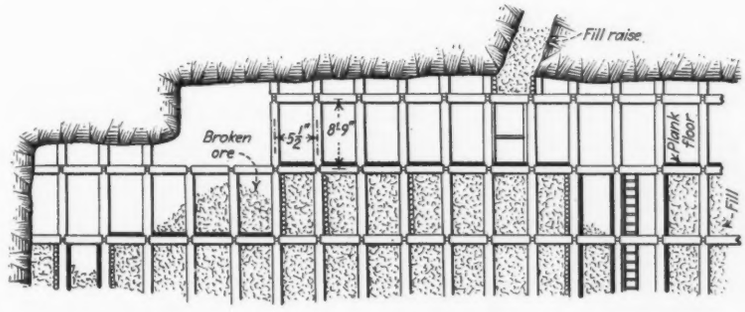


Fig. 2—A square-set stope, in longitudinal vertical section

like cross-sectional area is used to support the broken ore or waste fill. Round, unpeeled spruce poles, 4-5 in. top, are used as lagging. The type of timbering is largely that of posts and caps, but occasionally stulls hitched to the walls are used where the vein is narrow and the walls are good. It has been found best not to frame the caps tightly to the walls, but to allow a few inches to a foot leeway, this space being wedged tight with auxiliary blocking. When the walls are first open to the air, a preliminary squeeze seems to occur, probably due to air-slacking. Blocking takes up the strain without crushing the cap.

Chutes are placed at 16-ft. centers or in every fourth set. Eleven-foot lagging just fits this arrangement and also allows for flat handling on the main supply cage. The same chute is used in all types of stopes, the design having been unchanged for ten years.

Manways are spaced at 200-ft. intervals and are made of solid 6-8 in. cribbing 4 ft. 10 in. long and framed 5 in. deep by 6 in. on both ends. When a stope has been mined to within 50-60 ft. of the level above, a 48-deg. raise is put through to facilitate travel and handling of material.

All stoping is done by 3-in. and 3½-in. Leyner machines, drilling flat breast holes. A normal breast is carried the width of the ore and about 8-9 ft. high. Holes are drilled 10-12 ft. long, and care is taken to keep the side holes from running out into waste. Even with a level interval of 125 ft., much sub-drifting has to be done. This is common to any type of stoping, but with shrinkage it is often necessary to come down to the level and drive ahead and box hole to tap such an extension.

Cut-and-Fill Stoping—Chute spacing in cut-and-fill stopes has undergone much experimenting. Intervals of 35 to 150 ft. have been tried, and present practice has been made standard at 50 ft. with a manway alongside every third chute. Sawn half cribbing 8-9 in. in diameter is used for raising manways and mill holes. To raise a combined mill hole and manway one round takes the following timber: Three pieces 5 ft. 9 in. long framed 4½x7 in., one piece 5 ft. 9 in. long framed 9x7 in.; and three pieces 4 ft. 8 in. long framed 4½x7 in. To raise a mill hole alone per round requires four pieces 5 ft. 9 in. long framed 4½x7 in.

Chutes are lined with 3-in. spruce, and latterly 2- and 3-in. hardwood is being used. Landings are placed every 12 ft., and ladders, 2x4 in. with 1x3-in. rungs, are standard. For floors 2-in. hardwood or 3-in. spruce is used.

An 8-ft. breast is carried, 3½-in. Leyner machines drilling flat holes 10-12 ft. deep. In favorable ground the breast may be 50 ft. or more ahead of the muckers. On some of the narrower stopes (under 12 ft.) where the roof scales badly, the back is supported by timbers, locally called crown bars. At the time the breast is drilled, two holes about 4 ft. long are placed in each wall near the roof. Short starters are put in them, and the cross stulls blocked to the roof are supported on these "hitches." Where the stope is too wide for such support, and roof conditions are bad, recourse is had to an intermittent cut-and-fill; that is, one breast only is blasted, cleaned out, and the fill run in before the next breast is advanced.

Flat 1-ton stope cars of special design, or 1-ton Hudson type, are used with 16-lb. sectionalized rail (10-ft. lengths) to handle both ore and backfill.

On many of the quartz-type orebodies, scrapers are successfully used to slush the broken ore into the mill holes. The hoists vary in size from 7½ to 15 hp., and are all air-driven. Local conditions make use of electric scraper hoists inadvisable at present. The minimum efficient size for such a hoist is placed at 10 hp.

A stope crew usually consists of from four to eight men per shift—a machine man and helper, who drill, blast, scale, raise chutes, and the like; two to four muckers on ore; and two trammers on backfill. In the larger stopes a stope boss, who is paid at a higher rate, is in charge.

Square-Set and Fill — This is undoubtedly one of the safest and most flexible ways of mining heavy ground, and where extreme widths (up to 100 ft.) are encountered, it is the method used. The type of sets is standard, as listed in Table III.

Table III
Cap-Butting Type of Square Set

Cap, 10-11 in. round squared, two sides framed, 7 in. x 3½ x 5 ft. 6 in. long
Girt, squared, four sides framed, 7 in. x 9 in. x 4 ft. 11 in. long
Post, 10-11 in. round squared, two sides framed, 7 in. face x 1 in. x 8 ft. long.

Sets are designed to take lateral pressure, and where big widths are being mined, extensive use is made of timber cribs placed on the fill and blocked to the back as auxiliary support. When blasting, sets are reinforced by blast-

ing timbers dogged across the corners, and additional temporary posts support the caps. Breast holes, 10-12 ft. long, are drilled and blasted with a maximum of two to three sticks of powder per hole.

Chutes are spaced 50 ft. apart and are carried up in the regular sets and lined with 3-in. plank. Fill raises are 300 ft. apart, and ore and backfill are handled in 1-ton Hudson cars on 16-lb. sectionalized track, with turntables in place of switches.

Floor Pillars—Inasmuch as the haulageway on a level follows the vein for the most part, a passage must be maintained after the floor pillar has been mined. Square-setting has been found the most satisfactory way of doing this. All ore is recovered, and often there is more room on the level than previously. The practice is to leave a floor pillar of usually less than 20 ft., the thickness depending on width of vein and condition of roof and walls. Sills are laid on the existing fill and the sets are carried up through to the level. Temporary sills are laid down on the level floor and posted to the timbers or sets above, and not more than half of the area supported by these sills is mined before filling. In some cases, the posts supporting the level sets were cut off at the bottom and permanent sills slipped under and the square-sets carried through and blocked tight beneath and filled.

By this means, floors can be robbed under heavy ground or under old crushed timber. It is important to mine only a "bite" at a time and not to disturb the roof equilibrium.

Backfill — Waste rock is used exclusively for stope filling. There are five sources of supply: (1) Development and shaft sinking. (2) Sorting in stopes. (3) Old fill from finished stopes. (4) Waste crosscuts driven in walls of the stopes themselves. (5) Regular waste stopes. About 250 tons per day is obtained from the first three, leaving the fourth and fifth to supply the remainder. These are now discussed.

In certain areas, remote from waste passes, waste has had to be broken locally in stopes. Several schemes have been tried: Raises, inclined 25 to 50 deg., are driven into the walls, and the waste is either leveled by a scraper or shoveled or drawn into place from a chute. The practice now is to drive a crosscut 16 ft. into the wall and to advance and widen by slashing until the needed waste is obtained. As much as 2,000 tons has been broken in such a crosscut. Before the next slice is taken, the entrance is lagged off, and, with extra care in locating the crosscuts, an entire stope is filled without caving the walls. This is expensive and slows up stope work. During the past year it was necessary to break thus 19,940 tons of waste.

The following will illustrate the use of waste stopes: An area of about 100 by 200 ft. in soft altered porphyry was prepared for mining by caving. A large tonnage was obtained, but, with height, the ground was found to be silicified and broke in large blocks. The resultant necessary chute blasting decreased the daily tonnage. A similar scheme was tried with a shattered diabase dike about 15 ft. wide. A considerable tonnage was obtained, but the method was not entirely satisfactory.

A block of ground adjacent to the main waste pass system was therefore laid out as a sublevel stope and is now being mined. It is 270 ft. long, 110 ft. wide, and 175 ft. high.

A 9x9-ft. drift was driven through the center of the 270-ft. long block and a grizzly level established 30 ft. above the rail. Sublevels, 6½x5 ft., were driven from the main waste pass through and to the end of the area, 25 ft. apart c. to c., and a shrinkage stope 110 ft. long by 24 ft. wide by 147 ft. high was mined across the end. From that point on, all muck is obtained by slashing floor, roof, and walls of the sublevels. Estimate is

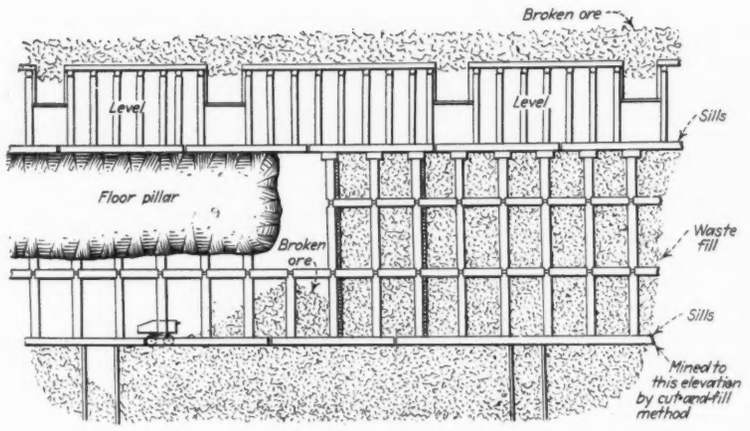


Fig. 3—Longitudinal section showing method of extracting floor sills

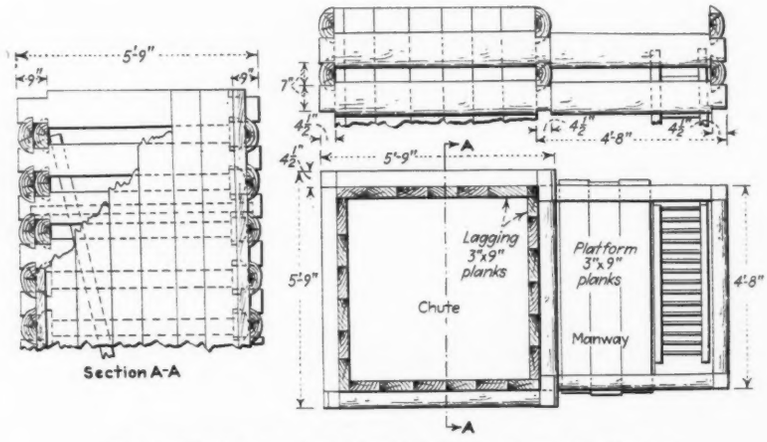


Fig. 4—Sketch showing general features of construction of combined chute and manway for cut-and-fill stope

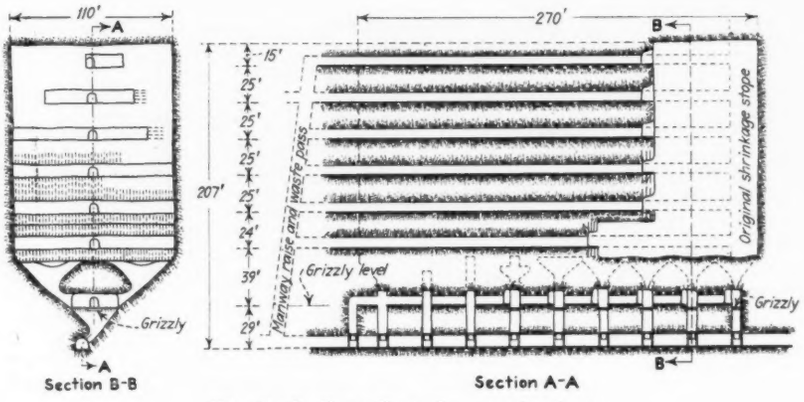


Fig. 5—Sections through a waste stope

In a square-set stope having crib support. One of the Hudson cars

Machine men take their explosive to the face in these bags, as described in text

made that 420,000 tons of waste will be broken here.

Ore and waste passes are driven at 55 deg. from the horizontal. In the former, control chutes are placed every 500 ft. vertically, whereas in the waste passes, a draw-off chute is built on each level as well as control chutes down to the 2,375-ft. level, so that waste can be either drawn from the pass or dumped into it on any level.

Ore from the west section of the mine is dumped into the ore passes feeding No. 6 shaft and is transferred on the 2,875-ft. level to No. 11 pass by fifteen-car trains of 2½-ton, Granby-type cars, hauled by 4-ton Goodman trolley locomotives. All openings are protected by dust-tight doors of special design, hand-operated by a lever-and-toggle arrangement, which locks in both open and shut positions. On the levels where Granby cars are used, similar doors are worked by remote-controlled air cylinders.

Underground Haulage — Grades are established at 0.5 per cent in favor of the load, with a minimum curvature of 30-ft. radius. Thirty-pound rail is used, the gage being 18 in. Frogs and switches are of the railroad type.

All cars have Timken bearings. Three types are in use: 19-cu.ft. end-dump; 20- and 30-cu.ft. Hudson-type; and 40-cu.ft. Granby.

The locomotives used include: two 4-ton double-motor trolley locomotives, each motor 20 hp.; six 2½-ton, 40-cell, worm-and-chain drive, storage-battery locomotives; two 1½-ton, single-motor, 3½-hp., storage-battery locomotives; and one 40-cell storage-battery Mancha.

On certain levels with isolated working places and long hauls, horses are used in place of locomotives. They are transferred from level to level, as needed, on No. 11 cage.

Blasting—Each miner does his own blasting. Forty per cent Polar Forcite and 55 per cent ammonia powder are used. In development headings the "cut" is blasted at least twice, usually the second time with the square up. Eight-foot 40-sec. black fuse is standard, with a No. 8 detonator. Fuse ends are dipped into a quick-drying paint of contrasting color (red or white), so that the miner, when trimming the fuse to obtain the proper rotation of firing, may more fully realize that any fuse end that lacks this mark has already been cut. Development crews are furnished with respirators. Air and water blasts are used in development work and stopes.

Certain types of work where measurement is possible are done on a bonus system. These include drifting, crosscutting, stoping, raising, timbering, and shaft sinking. Statements are rendered bi-monthly. Standard wages are guaranteed, and any bonus is divided equally among men participating. Powder is supplied by the company at cost and charged against the contract. Drifts and crosscuts are paid for at a standard rate per foot and timbering is paid for by the set placed. Stoping is subdivided into several operations, such as breaking, delivery, timbering, chute raising, and filling. Each class of work is paid for at a standard rate which varies with the width and type of work.

Tonnage and cost ratio of the various stoping methods for the past year are given in Table IV.

Table IV—Tonnage and Cost Ratio, Stoping Methods

	Per Cent Tonnage Stoped	Cost Ratio
Shrinkage	25	0.55
Cut-and-fill	41	1.00
Cut-and-fill with waste crosscuts	11	1.41
Square-set and fill	23	1.32
All stopes	100	1.00

Explosives—All explosives are stored half a mile from No. 11 shaft in five magazines with a capacity of 380 cases each. The magazines are arranged in a half circle and are connected by double air flues with a central heating unit in which are electric water heaters. These heaters supply radiators through which air is drawn by a fan to the flues of the various magazines. The central heater is thermostat-controlled.

Near the magazines is a small three-compartment building where capped

fuses and clay tamping cartridges are prepared. Two men, who also act as magazine guards, have charge of this building and make up the fuses and tamping sticks on contract. These are wrapped in blue paper to distinguish them from the powder.

Each day's individual orders for fuse and caps are made up in coils with the contractors' numbers attached and placed in a box container and transported by truck to No. 11 shaft. The day's powder order is also taken to the shaft by truck. There the powder is placed in red, wood-lined 20-cu.ft. steel cars having doors on one end and on top.

Guniting the walls of a crosscut. The men wear respirators

Underground, the day's powder supply is delivered in wood-lined steel cars, completely enclosed, to each contractor, and is placed in an individual, locked wooden box kept at a safe distance from the working place. The capped fuses for each contractor are placed in a locked fuse box kept 50 ft. from the powder box. These boxes are opened by the contractors and the powder and capped fuses carried to the face in separate rubberoid bags.

Dust Laying and Ventilation—In all blasting places compressed air-water sprays are used after blasting. These are obtained by the use of a cross-over hose connection joining the air and water lines of the working places. The first of these sprays was used in 1924 in a long drift from the end of a raise, to lay explosive fumes. It produced excellent results. Later, as the menace of fine dust was given more attention, the sprays were continued in use to bring down the dust, it being proved by actual dust counts that the air-water spray did this. The water is finely atomized by the compressed air, thus giving the fine dust particles numerous fine water particles to adhere to. As the air becomes super-saturated with water, the combined particles settle out.

Dust counts made of the dust from samples of air taken on the konimeter slide show that 20 min. after the blast of a cut in a heading, the air at the face for a distance of 30 ft. back (that is, to the end of air line) is clearer of the dust than before the blast, if an air-water spray has been used during that time. In the same face on the following day, using even a greater amount of air without water and a few minutes' longer time, the same area at the face showed a dust count many times higher, besides containing considerable fumes.

To carry a ventilation pipe nearer than 150-200 ft. to the face is not feasible, owing to the concussion and flying débris from the blast. In this ground, the whole round cannot be blasted at one time if long rounds are to be used. Returning to this face area is a source of dust danger for miners. So the air-water method described is used, and, in addition, the men wear respirators while walking back and forth to the face and while reloading. These respirators make a good extraction even on the heavy dust back from the face.

In stopes, drilling is done on both 7 o'clock shifts and blasted at 3 o'clock afternoon and morning without the miners returning for reblasting. An air-water spray of slightly different design is used for 4 hr. after blasting, leaving the atmosphere in the stopes not only clear of dust and fumes but with the fresh smell of water-washed air. If the previous shift does not turn on the water blast, the following shift reports it. For this reason it is always in constant use. To remove powder fumes and dust from headings, galvanized 12-in. diameter 18-gage and 16-in. diameter 16-gage pipe are used with Coppus fans at intervals as required, exhausting from a point 150-200 ft. back from face and delivering to the main air stream.

The mine is ventilated by natural ventilation through the eight shafts now open from the 200-ft. level to the 4,000-ft. level. The main air circuit is controlled by a fan on the 3,875 level. All shallow shafts are downcast, and the others either upcast or in a state of balance. This balance is maintained as closely as possible in working shafts which might freeze in winter if downcast. Only one working shaft, No. 6, is strongly upcast, and in it moisture from the rising air current cannot do injury of consequence. On the 3,875-ft. level at the bottom of No. 11 shaft, a 50-hp. Buffalo Forge fan delivers about 25,000 cu.ft. of air to the workings. At No. 12 shaft on the 3,875 level, about 12,000 ft. of the air stream is delivered to the bottom of the shaft and returned to the 3,750 backstream by a No. 425 Coppus fan.

Guniting—In April, 1927, guniting was first used underground at McIntyre Porcupine mines, to provide an air seal on exposed surfaces in No. 11 shaft stations and the deeper main crosscuts to prevent slacking and constant scaling. The general practice for air sealing is to give two light coats of gunite of $\frac{1}{8}$ to $\frac{1}{4}$ in. each. The surface to be sealed is carefully scaled to solid ground and washed with a combined air and water jet on one shift and the first coat applied on the next shift before the rock begins to loosen again. On the following day a second coat is applied. In this way about 100 ft. of 8x8-ft. crosscut per day is completed. The application in No. 12 shaft has already been described.

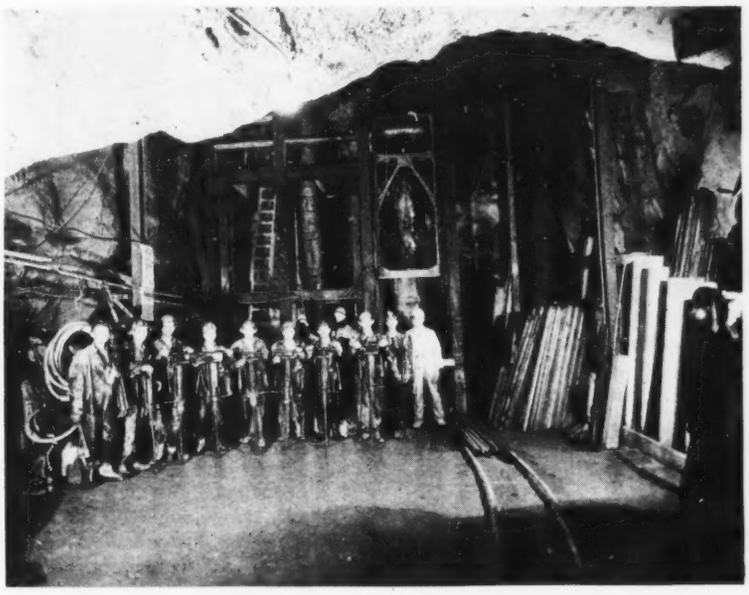
Treated Timber—Timber placed in the levels below the 3,000-ft. level is destroyed by fungus much more quickly than in the upper levels. The cost of replacing timber is a heavy item, increasing with depth. In gradually increasing amounts, timber treated with zinc chloride has been used in sizes up to 12x18 in. and has been found satisfactory to date, not only in preventing fungus growth but as added fire protection. At present hardwood elm stulls, posts, and plank are being prepared for use in levels below 4,000 ft. Elm, being a wood that takes treatment well and that is strong also, does not require as large sizes to hold the same load as do soft woods.

Drill Steel—Since 1922, 1-in. quarter-octagon hollow drill steel has been in use for all drills. A 0.8 carbon steel with a smooth borehole through the center is now almost standard. Canadian Atlas steel has been in use exclusively at McIntyre for six or seven years. (Details of the bits are given in another section.)

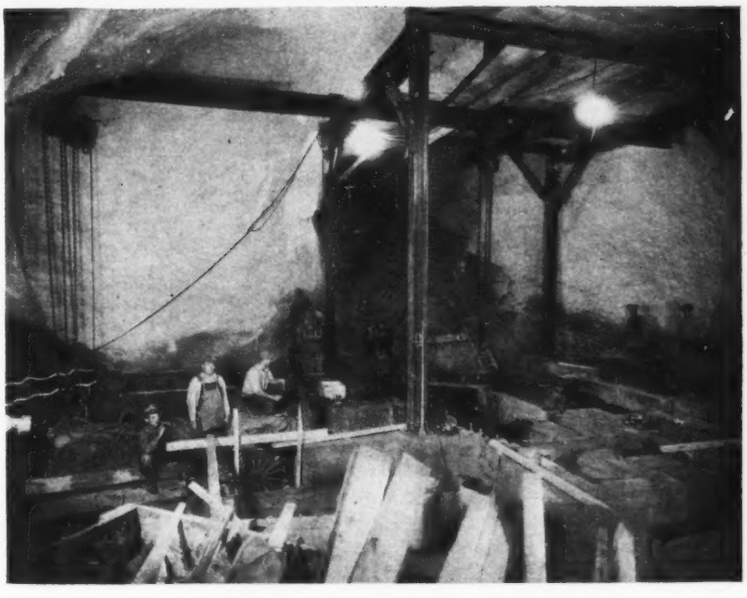
Deep-Level Development

The New No. 12 Sub-shaft

EARLY IN 1932 decision was made to sink an internal shaft, No. 12, below the deepest, or 3,875, level, to explore and develop veins known to extend below that horizon. A point 1,500 ft. south and 1,000 ft. east of No. 11 shaft was selected. This shaft was to start from a point almost three-quarters of a mile below the surface in a schistose porphyry mass, and would pass through the lava-porphyry contact at a depth approximately 5,000 ft. below the surface. It was, therefore, decided to sink a long, narrow shaft having its long axis at right angles to the strike of the schistosity. Shaft and equipment were planned to reach a depth of 3,000 ft. eventually, or practically 7,000 ft. below the surface. It was to be capable of hoisting 1,500 tons per day from that level, and to provide ventilation, compressed-air, water, and electric-power services to levels at 150-ft. intervals down to that depth. The 3,875-ft. level was selected for the main station and the 3,750 level for the hoist and dump.



In the 3,875-ft. level station at No. 12 sub-shaft. The sinking crew with plugger drills



Hoist room ¾ mile underground. Erecting No. 12 sub-shaft hoist

A four-compartment shaft was decided upon. Two hoisting compartments are 4 ft. 10¾ in. by 5 ft. ¾ in. inside dimensions. One compartment of the same size is lined and sealed for an airway, and a fourth compartment, 4 ft. 0¼ in. by 5 ft. ¾ in., is used for ladder, pipe, and cable way.

On Aug. 1, 1932, work was started in preparation. Crosscuts on the 3,750 and 3,875 levels from No. 11 shaft to the new shaft were first prepared for use; the 3,875 level as a permanent trolley haulage route and man travel way, and the 3,750 level as a minor haulage and service line way.

The station on the 3,875 level was cut 25 ft. wide, 45 ft. deep and 26 ft. high to serve the same uses as a surface shafthouse. On the level above, a hoist room, 55 ft. long, 45 ft. wide, and 24 ft. high, was excavated. The shaft was raised, full size, from the 3,875 level to 30 ft. above the 3,750 level, and, above this point, 16x12 ft. to the top, 93 ft. above the level. A rope raise 164 ft. long was driven from the hoist room to the sheaves located 78 ft. above the level. Ore and waste passes, corresponding to surface bins, were cut from the 3,875 to the 3,750 level. Rooms were also cut out on the latter level for transformers, switching equipment, and water-cooling pit for the hoist motor.

The crosscuts and the new openings, being in porphyry at depths where pressure effects are felt, were carefully scaled and then gunited to prevent further scaling and caving. The crosscuts were covered and sealed with a ¼-in. coat of gunite all over backs and side walls. Especially bad places received a second coat.

The ground in the 3,875-level station, in the 3,750-level hoist room, in the rope raise, and in the section of the shaft from the 3,875 level to the sheave station, scaled in places. From considerations of fire risk and permanency, timber was not to be used to support this ground, so reinforced guniting was employed. The reinforcing was discarded hoisting cable from which the grease had been burned, with woven No. 6 wire of 4-in. mesh, in shaft and stations, and the 1⅛-in. cable with sheets of expanded metal in the hoist room. The backs were reinforced and gunited while the broken muck was close to them.

Many thin coats were applied, until the reinforcing was completely covered. The gunite in and over the reinforcing was 6 in. thick in places. The side walls were similarly gunited, but in heavier coats, in sections 12 ft. high, as the broken muck was removed. The last coat of gunite was rubbed down smooth with a kitchen broom while fairly wet, and on this surface a coating of lime putty was sprayed by a whitewashing machine.

So firmly and tightly was the reinforcing applied to the walls that it did

not sag during the guniting, with the result that the gunite is firmly bonded to the back and walls.

Shaft Timbering—Steel framing was used in the shaft-head works above the 3,750-ft. level for skip dumps, guide holders, and sheave-wheel supports. Below the level the shaft timber, 8x8 in. dressed ¼-in. scant, is used except for a 26-ft. steel shaft set at the 3,875 level. It is British Columbia Douglas fir and is treated with ½ to ¾ lb. of zinc chloride per cubic foot. This not only protects the timber against rot and fungus growth but also has a definite fire-resisting quality. Timber was all framed before treatment.

Inasmuch as it is not possible to have all blocking treated after being cut, eastern cedar is used for blocking, packing and wedges. This resists decay but is soft. Wedging has therefore to be done with more than ordinary care. In sinking, the overbreak has been such that less cedar packing than expected has so far been used.

Shaft timber is lined around its whole circumference with 2-in. British Columbia fir, treated with zinc chloride. The sets are 7 ft. center to center.

Hoisting compartments Nos. 1 and 2 are fitted with guides, 28 ft. long and of 7¾x5¾-in. British Columbia fir. They are bolted to steel brackets having ⅝-in. shoulders each side of the guide to prevent splitting. The brackets are bolted to the shaft end plates and dividers.

No. 3 airway compartment is lined with 1-in. Douglas fir, also treated with zinc chloride. Over this 4x5-ft. galvanized iron sheets, lapped 2½ in., are laid. Two sheets are turned up 1¾ in. and two sheets 3¾ in., making a good lap for corners. All laps are given a band of Tremco caulking compound. Straight laps are nailed down with two rows of 1¼-in. braided galvanized nails. Corner laps are fastened by a 3-in. triangular strip of Douglas fir nailed with 5-in. No. 8 gage nails. The lining of this compartment cannot be completed until all blasting has been finished near the shaft, and in the meantime 20-in. galvanized pipe is used for ventilation.

No. 4 compartment has a ladderway with platforms every fourteen feet. Around two sides of the ladders is an 8-in. space for 6-in. Victaulic-coupled air line, 3-in. water line with reducing valves every 250 ft., 3-in. diameter pump line, and electric cables.

Sinking equipment consists of two buckets 4 ft. high having 29-cu.ft. capacity. These are used in the No. 2 hoisting compartment and have double chain bails, any one chain being strong enough to carry the load. The buckets are hung between angle-iron guides and from the bottom of an aluminum crosshead, which is equipped with release dogs to allow the bucket to be moved either to the bottom of the shaft or to its dumping position at the collar. The crosshead has a flat, hardwood roof and can carry the whole shaft crew at one trip. At the top of the buckets are dumped on to a door which deflects the muck into the waste pass. This door fits the walls closely to prevent spill.

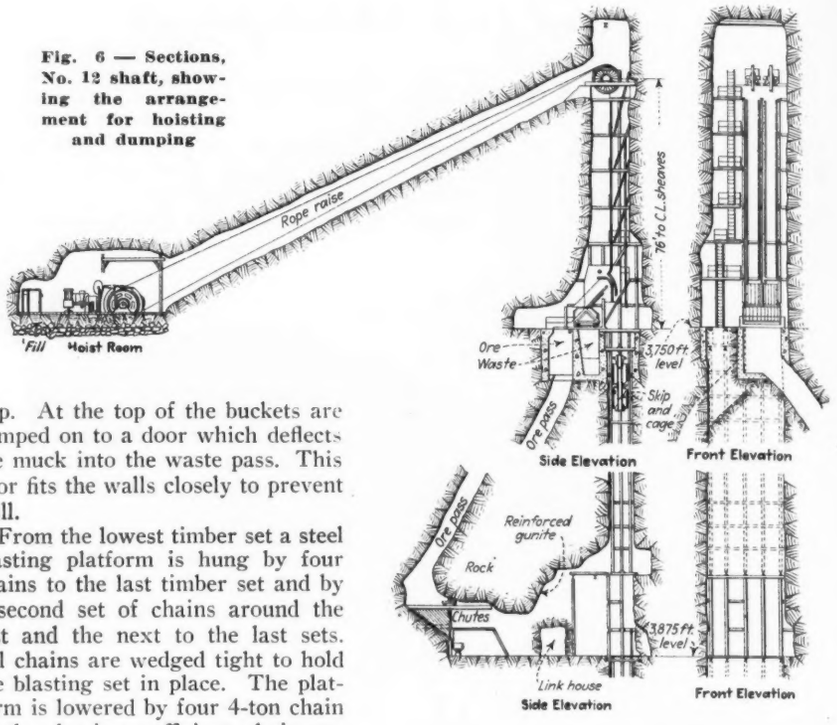


Fig. 6 — Sections, No. 12 shaft, showing the arrangement for hoisting and dumping

From the lowest timber set a steel blasting platform is hung by four chains to the last timber set and by a second set of chains around the last and the next to the last sets. All chains are wedged tight to hold the blasting set in place. The platform is lowered by four 4-ton chain blocks, having sufficient chain capacity to lower the blasting set 48 ft. From this set to the bottom of the shaft a chain ladder is used.

An air and water manifold, connected to the permanent air and water lines by 50-ft. lengths of air and water hose, is hung in No. 1 hoisting compartment from a small hoist. A cluster of electric lights is also lowered.

Air is delivered to the men through 20-in. galvanized-iron pipe connected to a No. 425 Coppus fan. A small exhaust fan is used for half an hour after blasting, and compressed air and water are blown into the bottom of the shaft.

Permanent hoisting equipment will include two combined cages and 3-ton skips, to be hoisted in balance by a Fraser & Chalmers hoist which is now in operation. The hoist has 8-ft. drums

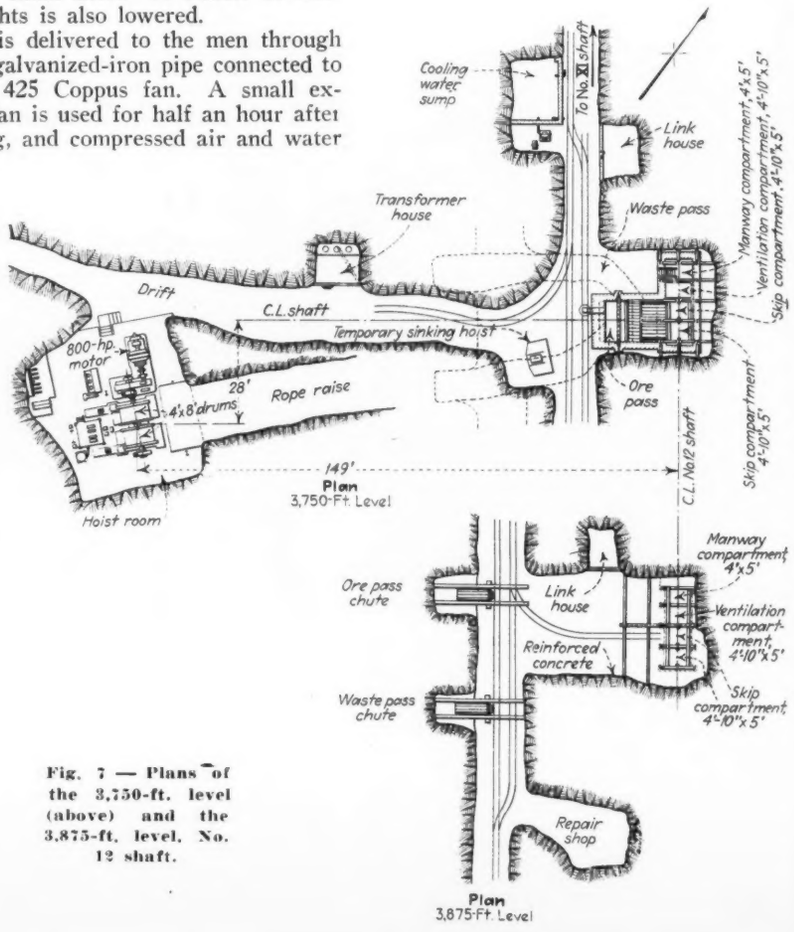
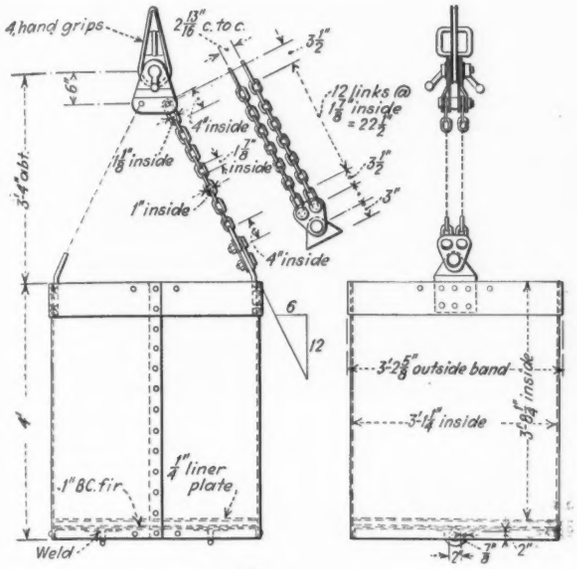
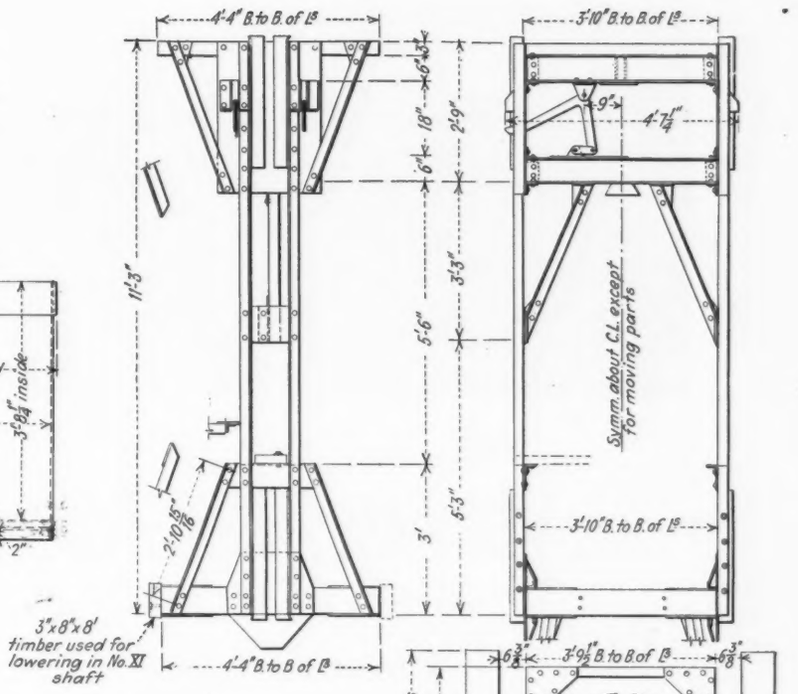
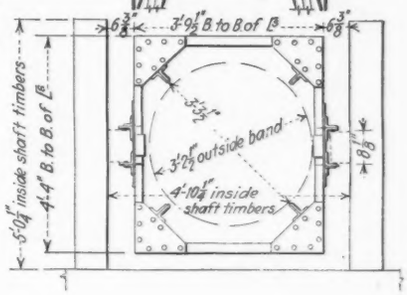


Fig. 7 — Plans of the 3,750-ft. level (above) and the 3,875-ft. level, No. 12 shaft.

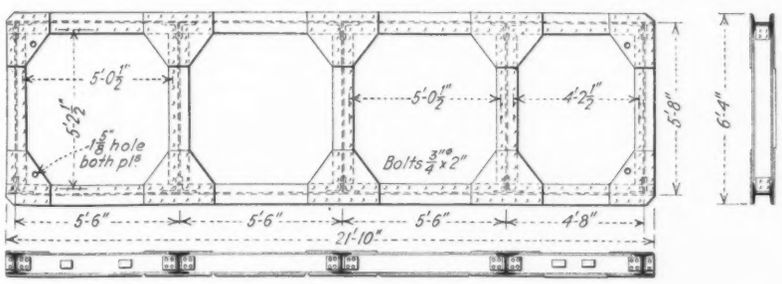






Left: Fig. 8—Sinking bucket, No. 12 shaft.

Right: Fig. 9—Sinking crosshead, of aluminum, used in No. 12 shaft.



Left: Fig. 10—Blasting set, of steel, that is now in service in No. 12 shaft.

4 ft. wide and is driven by an 800-hp. motor. The rope is 1⅛ in. in diameter and rope speed 1,500 ft. per minute.

Actual sinking below the 3,875 level was started through No. 2 compartment on April 3, 1933. To date the average monthly advance has been 200 ft. in the shaft, one 32-ft. station, complete with shaft timber, and one set of five 8x16-in. bearer timbers placed 14 ft. below the station levels, which are 150 ft. apart. For bonus purposes, station footage is considered the same as shaft footage. On this basis 250 ft. was advanced in August.

The shaft-sinking round has 54 9-ft. holes, with a center V-cut. Fifty sets of 1-in. quarter-octagon steel are used. Four steels comprise a set, the longest being 10 ft. Drills are six Rand No. 82 sinkers fitted with handles, foot rest, and blower. One deck man goes down to help drill, and a helper is assigned to each two machines. Holes are drilled in pairs, and as a steel is finished the drill is lifted from one hole to the other, the helper having placed a new steel in the meantime. In this way, the drill runs almost continuously and six machines can drill 486 ft. in 2 hr. on an average.

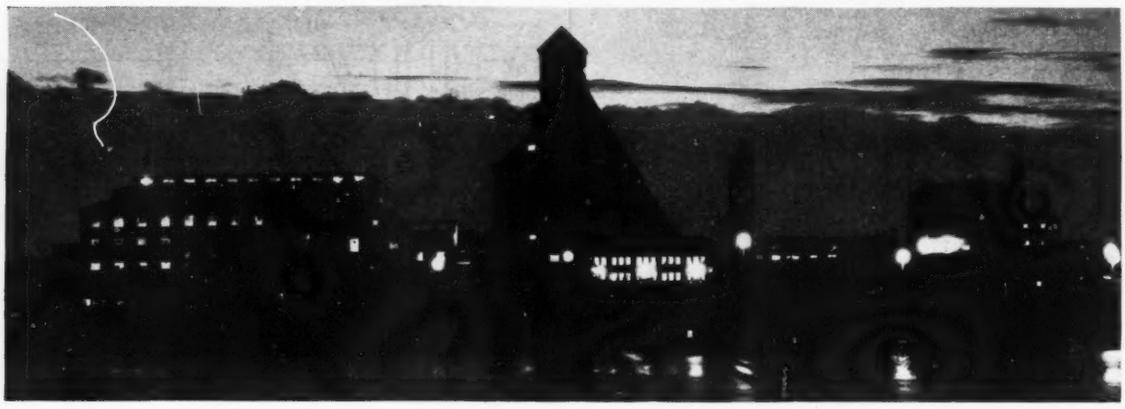
In blasting, twelve cut holes are shot and mucked; then a second bank of holes is shot and mucked. During this mucking a center 7-ft. cut of four holes is drilled. This and the final third bank of the round are then shot and mucked. The four-hole cut serves as a water collector and as a start for the cut of the next round. Blasting is done with electric time fuses fired from the lighting current, special safety precautions being used.

Timber is customarily placed three sets at a time, two hours being needed to place and block a set. Sinking compartments are lined, and, if necessary for wall support, are packed with 6-ft. cedar. The other two compartments are completed by four timber men, who work day shift only.

With this routine 40 men average about 8 ft. of completed shaft a day. The crew on each of the three shifts consists of one hoist man, two deck men for the two levels (3,875 and 3,750), eight miners, one leader, a captain, and four extra timber men.

The shaft bottom is now over 5,000 ft. below surface, and no undue pressure or other feature has appeared that might make mining difficult. In fact, the ground in the last station gave less trouble than did the 3,750 and 3,875 levels, where quartz in the areas to be excavated made the walls bad. This was apparently only a local condition.

Work to date indicates that mining to a depth of 7,000 ft. will not present unusual difficulties, either because of the nature of the ground or of high temperatures.



McIntyre Metallurgy

Flotation and Cyanidation

J. J. Denny

MILLING OPERATIONS started at the McIntyre Porcupine property in March, 1912, with a ten-stamp mill embodying amalgamation and table concentration. This plant, having a capacity of 50 tons, was designed and erected by R. J. Ennis to treat the ores mined at or near the surface, in which fine and coarse gold predominated with the quartz. By this method of treatment a recovery of better than 90 per cent of the values was obtained, amalgamation accounting for 80 per cent of the gold.

However, with deeper development, the character of the ore changed, less gold was found in the quartz, and more in the sulphides, a condition not favorable to amalgamation. After several months' operation, experiments by A. Dorfman[1] demonstrated the logical method of treatment for this class of ore to be fine grinding and cyanidation. In 1913, the amalgamation and concentration plant was replaced by a 150-ton, all-sliming, counter-current decantation plant. From then on, similar units were added to accommodate the increasing tonnage, until, in 1925, bowl classifiers and American filters were installed, the former to take care of the increasing amount of auriferous sulphides, which required finer grinding, the latter to reduce the soluble gold and mechanical cyanide losses from the counter-current decantation process.

With the constant growth of the mine, it became apparent in 1928 that the old plant was no longer adequate, and to treat economically an additional 700 or 800 tons daily, it would have to be remodeled or entirely replaced. As mining operations were being centralized at No. 11 shaft, decision was made to built an entirely new plant at that point. The question then arose as to the type of mill most suitable for treating the ore. Table concentration, followed by cyanidation of the concentrates, was considered, but experiments indicated that, although a small tonnage of high-grade concentrate could be recovered, a table tailing below the economic limit of cyanidation could not be obtained. Flotation suggested itself, as this process had reached a high state of efficiency in the treatment of base-metal ores, due in part to improvement in flotation machines, together with the discovery of new promoting and frothing reagents, having a greater selectivity for mineral particles than the coal-tar and creosote products previously used. Other features which influenced the choice of flotation were: less capital expenditure for building and equipment; a smaller plant to be heated and lighted; and lower operating, depreciation, and maintenance costs.

Preliminary laboratory experiments indicated the feasibility of the process as applied to McIntyre ore and the justification of installing a 150-ton pilot plant. Results obtained from this unit during twelve months of operation confirmed the initial tests.

Excavations for the new flotation and cyanidation plant started July 28, 1930, and metallurgical operations began May 27, 1931. During the tuning up of the new plant, the old cyanide mill continued in operation on a reduced tonnage, and finally closed down in November, when the new plant had been brought up to its rated capacity of 2,000 tons daily.

McIntyre ore consists of quartz, porphyry, and schistose basalt and dacite, with a pyrite content of 3 to 15 per cent. The gold is associated with both the quartz and sulphides, predominating to a greater extent in the latter. A typical analysis of the ore fed to the mill over a period of 14 months is given in Table I.

The general scheme of treatment consists of crushing the run-of-mine ore in jaw and cone crushers to 1 in., and reducing the crusher product to ⅛ in. by rolls in closed circuit with vibrating screens. The screen product is ground to minus 65 mesh in tube mills operating in closed circuit with single flotation cells and classifiers. Classifier overflow is floated, with elimination of a finished tailing. Concentrate from the flotation cells is reground in cyanide solution to minus 325 mesh in tube mills, operating in closed circuit

Table I—Average Analysis, Mill Feed, Fourteen Months

	Per Cent
Iron pyrite	8.21
Silica	54.92
Combined water and carbon dioxide	4.32
Ferric oxide	6.28
Aluminum oxide	14.18
Calcium oxide	4.90
Magnesium oxide	4.33
Potassium and sodium oxide	2.85
Gold, per ton	$7.70
Specific gravity of dry ore	2.84

[1] *Transactions* of the Canadian Mining Institute, 1917.

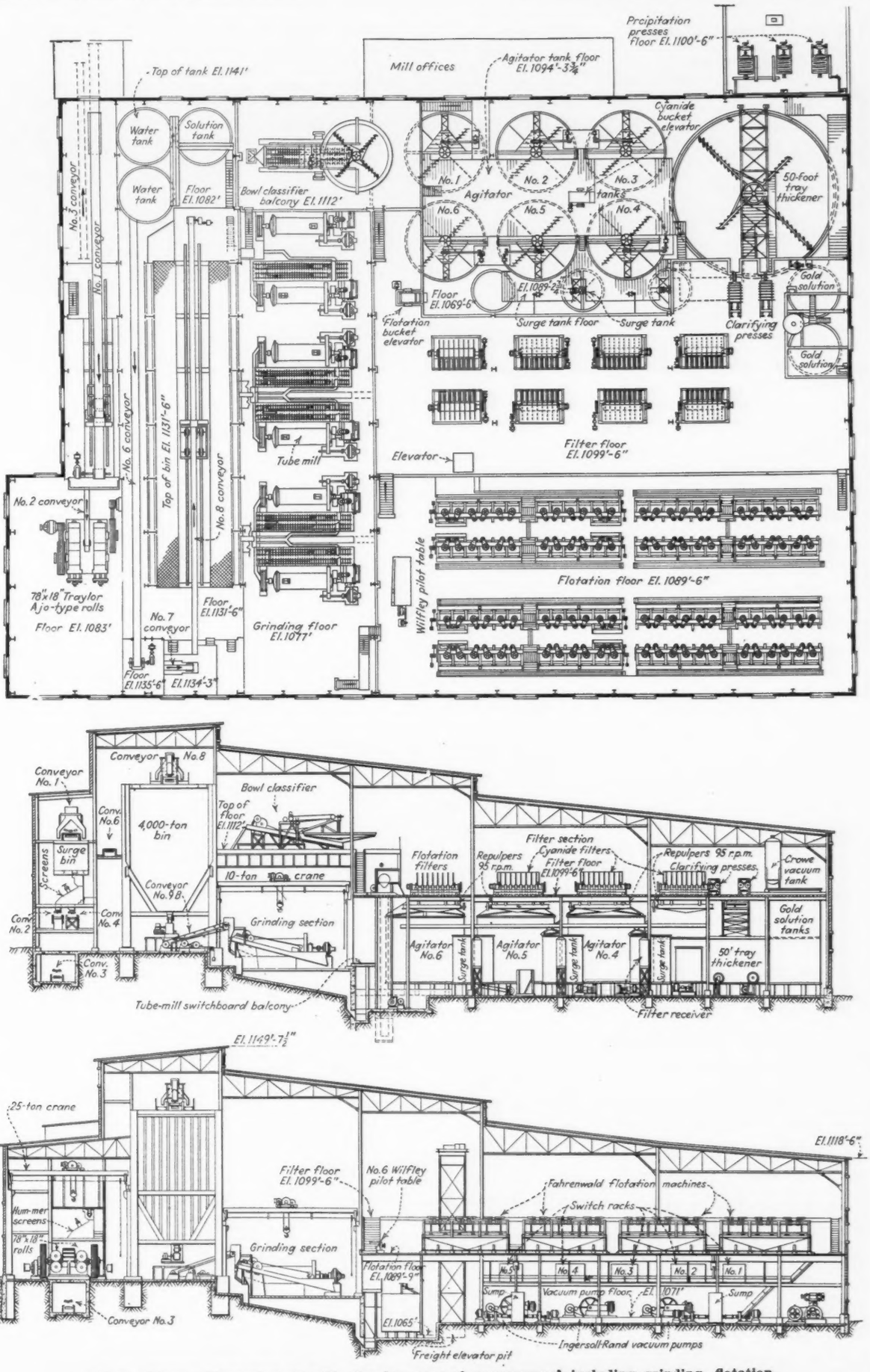


Fig. 1—Above—Floor plan of mill, showing general arrangement including grinding, flotation, and cyanidation. In the center—Sectional elevation of mill between Columns 1 and 7, looking east and showing the cyanidation section. Bottom—Similar sectional elevation between Columns 7 and 10, looking east

Legend for Flowsheet, Fig. 2

Crushing Section

1. Underground Traylor jaw crusher, 36x48 in.
2. No. 11 shaft
3. Receiving bin, 750 tons
4. Pan conveyor
5. Magnet, No. 5 size
6. Symons cone crusher
7. No. 1 conveyor
8. Magnet, No. 4 size
9. Merrick Weightometer
10. Automatic tripper
11. Surge bin, 190 tons' capacity
12. Six 6x4-in. Hum-mer screens
13. No. 2 conveyor
14. Traylor Ajo-type rolls, 78x18 in.
15. No. 3 conveyor
16. No. 4 conveyor
17. No. 5 conveyor
18. No. 6 conveyor
19. No. 7 conveyor
20. No. 8 conveyor
21. Automatic tripper
22. Sly dust collector

Flotation Section

23. Mill ore bin, 4,200 tons
24. Five conveyors, 30 in.
25. Conveyors Nos. 9A, 9B, and 9C
26. Five Allis-Chalmers 5x16-ft. tube mills
27. Five combination flotation and hydraulic classification machines
28. Five Dorr classifiers
29. Morris pump, 6 in.
30. Pulp distributor
31. Eight primary flotation cell units (No. 24 Denver Fahrenwald Sub-A)
32. Morris pump, 6 in.
33. Pulp distributor
34. Six secondary flotation cells (No. 24 Denver Fahrenwald Sub-A)
35. Morris pump, 6 in.
36. Morris pump, 2 in.
37. Wilfley pilot table
38. Wilfley pump, 2 in.
39. Automatic sampler
40. Dewatering filter surge tank
41. Wilfley pump, 4 in.
42. Three dewatering American filters

Cyanidation Section

43. Three repulpers
44. Two Allis-Chalmers 5x16-ft. tube mills
45. Two 30x6-ft. Dorr classifiers
46. Morris pump, 6 in.
47. Six 24x20-ft. Dorr agitation tanks
48. Morris pump, 6 in.
49. Dorr bowl classifier, 30x6ft.x20 ft. bowl
50. Dorr tray thickener, 50x14 ft.
51. Two Dorrco duplex diaphragm pumps
52. Wilfley pump, 4 in.
53. Two first-stage American filters
54. Two repulpers
55. Surge tank
56. Wilfley pump, 4 in.
57. Two second-stage American filters
58. Two repulpers
59. Surge tank
60. Wilfley pump, 4 in.
61. One third-stage American filter
62. One repulper
63. Wilfley pump, 6 in.
64. Two cyanide-section flotation units
65. Cyanide-cell concentrate to grinding circuit
66. Cyanide-cell tails to sampler
67. Morris pump, 6 in.
68. Pregnant solution (unclarified) storage tank, 15x20 ft.
69. Aldrich triplex pump, 7x9-in.
70. Two Merrill clarifying presses, 42 in.
71. Storage tank for clarified pregnant solution
72. Roturbo centrifugal pump, 4 in.
73. Crowe vacuum receiver
74. Aldrich triplex pump, 7x9 in.
75. Three Merrill precipitation presses, 52 in.
76. Aldrich triplex pump, 7x9 in., barren solution
77. Tank, first-stage filtrate
78. Roturbo centrifugal pump, 4 in.
79. Tank, second- and third-stage filtrate
80. Roturbo centrifugal pump, 6-in.
81. Aldrich triplex pump, 7x9-in.
82. Dewatering filtrate receiving tank
83. Roturbo centrifugal pump, 4 in.
84. Fresh water storage tanks, 16x24-ft.
85. Cyanide-solution storage tank, 16x24 ft.
86. Three 23x12-in. dry vacuum pumps
87. Alley-McLellan compressor, 2,200 cu.ft.
88. Sullivan compressor (spare), 1,000 cu.ft.
89. Precipitate to refinery

Refinery

90. Acid-treating tank
91. Montejus
92. Perrin filter press, 30 in.
93. Fluxing pan
94. Two No. 2 Rockwell smelting furnaces
95. Refined bullion to Mint
96. Slag to smelter

with classifiers. Next comes agitation of the pulp and thickening of it, followed by three stages of filtering and washing. Final residue goes to waste. Precipitation of the pregnant solution is done by the Merrill-Crowe system. The precipitate goes to the refinery and the gold bullion to the Mint.

A direct comparison of operating costs, together with residue values, between the operation in the two final years in the old mill and the last sixteen months in the new plant, shows a saving of 12.1c. in costs, and a decrease of 15c. per ton in the residue, or a total difference of 27.1c. per ton, in favor of the new mill.

From the beginning of operations in the new plant, the costs and over-all extraction have been up to expectations, and the results have fully justified the adoption of the combination flotation and cyanidation process.

I take pleasure in acknowledging my gratitude to the mill staff for their cooperation during the development and operation of the process now to be described.

Details of the various units follow:

Underground Preparation — Run-of-mine ore is fed from the ore pass on the 3,875-ft. level to a jaw crusher by a 64-in. Ross chain feeder, the chain of which weighs 11,100 lb. This is driven by a 5-hp. motor through a bronze worm reducer, with a sprocket-chain-gear-and-pinion arrangement, giving a speed of 19 r.p.m. and a capacity of 175 tons per hour. The jaw crusher, a 36x48-in. Traylor Blake-type unit, is driven at 140 r.p.m. by a 150-hp., 2,200-volt, slip-ring motor, through a V-belt drive. Its opening is 6 to 8 in. and the stroke 2 in., at which setting the capacity is 170 tons per hour, from run-of-mine to 7-in. ring. From the crusher the ore drops to a loading pocket, from

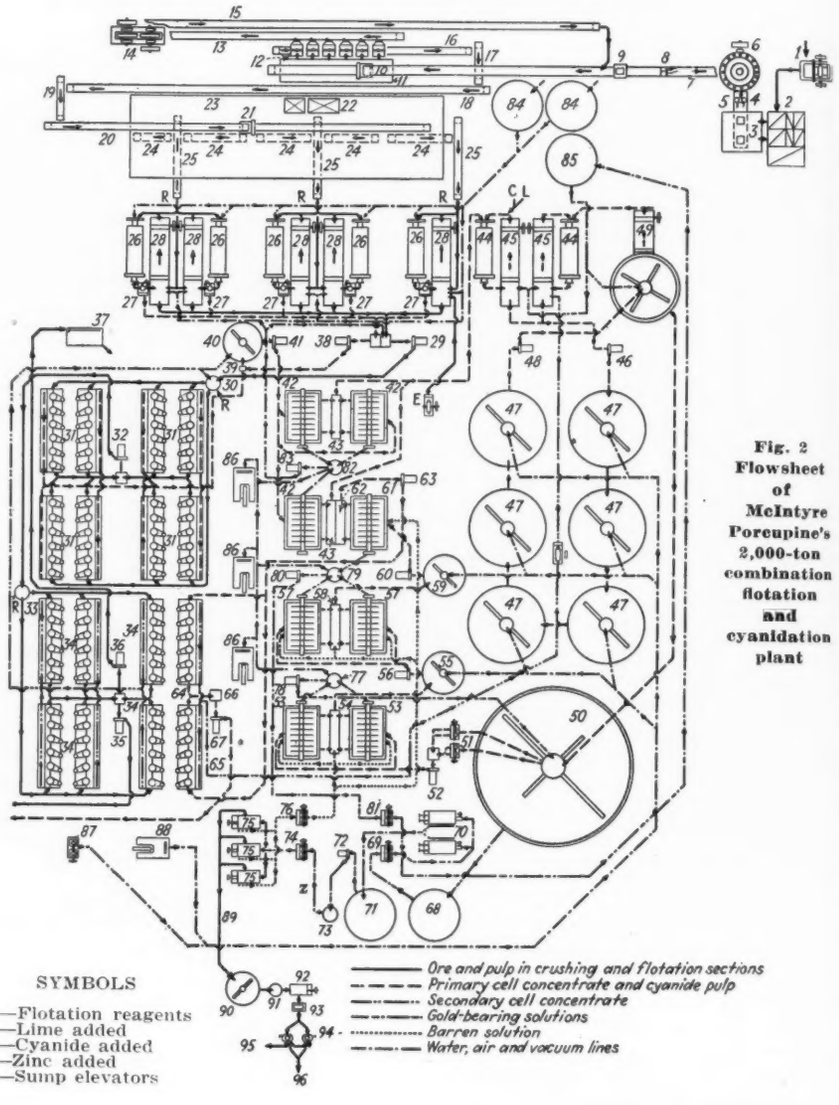


Fig. 2 Flowsheet of McIntyre Porcupine's 2,000-ton combination flotation and cyanidation plant

Table II—Total Operating Costs

	Per Ton
Crushing and conveying	$0.1070
Flotation	0.3633
Cyanidation	0.2734
Refining	0.0222
Assaying	0.0152
Mill alterations	0.0041
	$0.7852

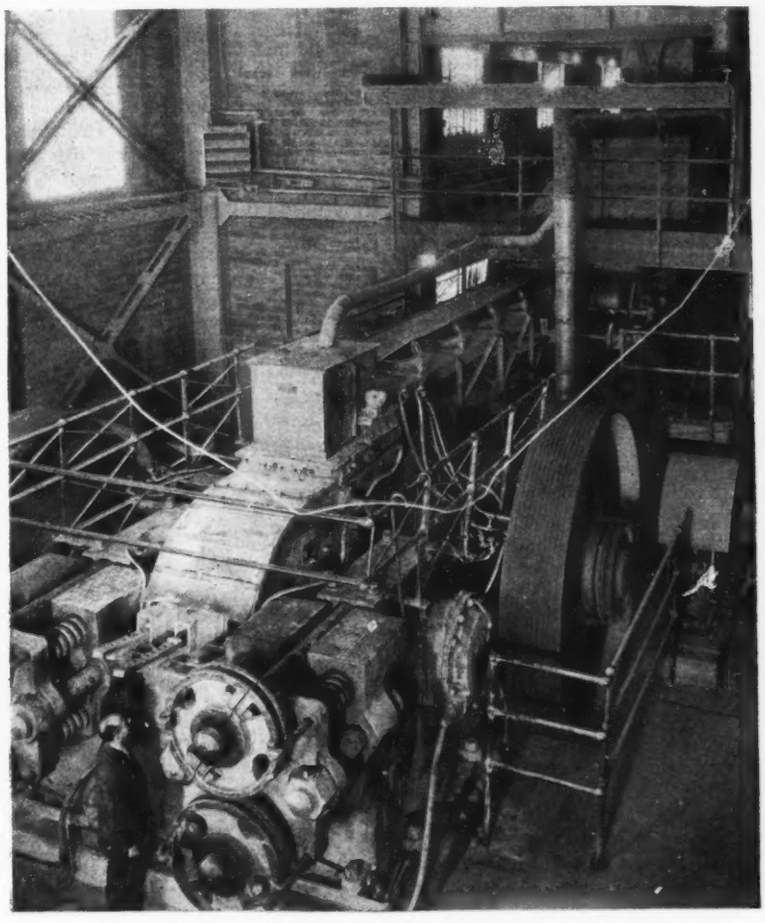
which it is hoisted to surface in two 6-ton skips, and dumped into a 750-ton steel storage bin, above the cone crusher.

Surface Crushing—From the storage bin, the ore is fed through two chutes, equipped with finger gates, to a 4x26-ft. (centers) pan conveyor, driven by a 7½-hp. motor through a 22:1 worm gear reducer. Speed of the conveyor is controlled by variable-speed reducer, giving a range of 4 to 20 ft. per minute. At 15 ft. per minute, capacity is 150 tons per hour. This discharges the ore to a 7-ft. Symons cone crusher.

tramp iron. Its coil capaciy is 5,300 watts cold and 3,480 watts hot. It is operated by a generator set developing 9½ kw.

For further safeguard, a second magnet, 45 in. in diameter, is hung over No. 1 conveyor near the cone crusher discharge. The coil is excited by a generator set developing 7½ kw. at 250 volts, and is directly driven by a 3-hp., 550-volt motor.

No. 1 conveyor transfers the ore from the cone crusher to the surge bin, and is in closed circuit with the rolls.



Ajo-type rolls, 78x18 in., crushing from 1 in. to 3/16 in.

This crusher is set at $\frac{7}{16}$ in. and is driven at 440 r.p.m. by a 200-hp., 2,200-volt, slip-ring, dust-proof motor, through a V-belt drive. It is equipped with a De Laval oil pump and purifier system. It reduces the 7-in. product to approximately 1 in. at the rate of 150 tons per hour. Liners are of manganese steel, 2¼ in. thick, and last ten months. Due to the breaking down of the zinc, back of the mantle and bowl liner, this metal has been replaced by babbitt, because of the deleterious effect of the zinc should it find its way into the cyanide circuit.

A 55-in.-diameter magnet weighing 3¼ tons is suspended over the ore stream feeding the crusher, to take care of

The belt is 36 in. wide by 230 ft. long, c. to c., the first 164 ft. being inclined at 20 deg. to the horizontal and the rest being level over the surge bin. It is driven at 275 ft. per minute by a 40-hp., squirrel-cage, dust-proof motor through a Falk speed reducer. It has a capacity of 400 tons per hour, handling the current ore as well as a circulating load of 190 per cent from the rolls.

On No. 1 conveyor a Model "E" Merrick Weightometer is installed between the cone crusher and the point where the return product from the rolls joins the current flow. This has a capacity of 190 tons per hour. It weighs all incoming ore.

A Link-Belt tripper automatically discharges the ore from No. 1 conveyor to the surge bin through two chutes. It is driven at 15 ft. per min. by a 10-hp. motor. The longitudinal openings at the top of the surge bin are covered with strips of rubber belting, hinged to the side to confine dust. A plough arrangement, attached to the tripper, lifts the belting clear of the distributing chutes, allowing it to fall back into place as the tripper passes.

The surge bin is of steel construction, 48 ft. long, 10 ft. wide and 8 ft. deep and holds 190 tons. It is above the vibrating screen unit, the ore being discharged through six chutes equipped with radial gates and drum feeders, 3 ft. 8¾ in. long by 18 in. in diameter, attached to a continuous shaft, directly driven at 10 r.p.m. by a 7½-hp., squirrel-cage, dust-proof motor, through a reducer.

From the surge bin the feed passes over six 4x6-ft. single-deck Hum-mer electric vibrating screens, set at 33½ deg. to the horizontal. These are equipped with vibrators excited by a generator, developing 109 amp. at 110 volts, 15 cycles. The generator is driven by a 10-hp., 550-volt, squirrel-cage motor. The screen cloth is of special wire with $\frac{3}{16}$x$\frac{5}{8}$-in. openings and has a life of approximately 70 days.

Screen oversize is conveyed to the rolls by No. 2 conveyor, which is horizontal and of the shuttle type, so that it may be moved back when repairing the rolls. It is 24 in. wide and 70 ft. c. to c., and is driven by a 5-hp., internal-geared 6:1 ratio, squirrel-cage, dust-proof motor, through a V-belt drive, giving a speed of 275 ft. per min., with a capacity of 400 tons per hour.

One set of 78 in. (diameter) by 18 in. Traylor (Ajo type) rolls reduces all screen oversize to $\frac{3}{16}$ in. The rolls are choke-fed, and are set at $\frac{1}{8}$ in. Each is driven at 125 r.p.m. by a 150-hp., 2,200-volt, slip-ring motor, through V belts, giving a peripheral speed of 2,550 ft. per min., with new shells. Spring pressure on the roll face is 40,000 lb. per lineal inch.

Hadfield chrome shells 8$\frac{11}{16}$ in. thick are used. They have a life of approximately 95 days and wear to 2½ in. thick before slippage necessitates their removal. They wear in the center without grooving. To maintain an even face emery stones are imposed on the edges. The rolls have a gravity oiling system and a Bowser oil filter.

After passing through the rolls, the ore is returned by No. 3 conveyor to the transfer point at No. 1. No. 3 conveyor is 24 in. wide and 178 ft. c. to c., and is set on a slope which gradually increases from 1 deg. at the feed end to 12 deg. for the last 89 ft. It has a capacity of 260 tons per hour, and is driven at 275 ft. per min. by a 20-hp., squirrel-cage, dust-proof motor, through a 15.7:1 Falk reducer.

From the screens, the $\frac{3}{16}$-in. finished product drops directly to No. 4 con-

Table III—Screen Analysis of Crusher House Products

Conditions — No. 7 fine Symons cone crusher. No. 390 Ton-Cap screen cloth on Hum-mer screens. Rolls set to give a 3/16-in. product. Tonnage crushed — 2,100 tons in 16 hr.

Mesh	Symons Feed	Symons Discharge	Screen Feed	Roll Feed	Roll Discharge	Mill-Bin Feed
Plus 6-in. ring	16.1%	.....	.....	.....	.....	.....
Plus 4-in. ring	7.8	.....	.....	.....	.....	.....
Plus 2-in. ring	17.5	.....	.....	.....	.....	.....
Plus 1½-in ring	5.7	.....	.....	.....	.....	.....
Plus 1.050 (Tyler)	3.4	1.5%	0.8%	1.3%	0.3%	.....
Plus 0.742	7.1	8.7	4.2	6.8	1.9	.....
Plus 0.525	6.4	17.2	7.3	10.4	2.9	.....
Plus 0.371	5.1	16.3	8.3	13.5	5.1	.....
Plus 3 Mesh	4.4	9.8	9.8	16.1	11.9	.....
Plus 4	3.4	6.8	20.6	31.0	29.0	2.2%
Plus 6	3.4	7.1	16.5	16.6	20.7	14.6
Plus 8	1.8	4.2	8.7	2.1	6.2	15.7
Plus 10	2.0	5.2	4.9	0.5	5.6	15.2
Plus 14	2.2	2.0	2.9	0.2	2.7	8.6
Plus 20	1.5	2.9	2.6	0.1	2.5	8.1
Plus 28	1.7	2.7	2.1		2.2	6.6
Plus 35	1.4	2.7	1.8		1.7	5.5
Plus 48	1.4	1.8	1.4		1.3	4.3
Plus 65	1.0	1.8	1.3	0.5	0.9	3.1
Plus 100	1.2	1.5	1.1		0.9	3.0
Plus 150	0.7	1.0	0.8		0.7	1.9
Plus 200	0.4	0.7	0.6		0.4	1.7
Minus 200	4.4	6.1	4.3	0.9	3.1	9.5
	100.0	100.0	100.0	100.0	100.0	100.0

veyor, 24 in. wide and 101 ft. c. to c., the last 41 ft. being inclined 5 deg. This is driven at 200 ft. per min. by a 5-hp., squirrel-cage, dust-proof motor, through a 21.3:1 Falk reducer. The belt has a capacity of 190 tons per hour, and discharges to No. 5 conveyor.

The latter is a short belt 24 in. wide with 11 ft. centers, inclined at 18 deg. It is operated at right angles from No. 4, discharging to No. 6 conveyor.

No. 6, which receives ore from No. 5, is 30 in. wide and 176 ft. c. to c., the first 115 ft. being inclined at 19½ deg. and the remainder at 13 deg. It is driven at 215 ft. per min. by a 20-hp., squirrel-cage, dust-proof motor, directly connected through a 29:1 Falk reducer, and having a capacity of 190 tons per hour. Discharge goes from here to No. 7 conveyor at the top of the mill storage bin.

No. 7 is 24 in. wide and 16 ft. c. to c. and is inclined at 19 deg. and operated at right angles to Nos. 6 and 8. It has the same drive, speed, and capacity as Nos. 4 and 5, the product being conveyed to No. 8. The latter is 24 in. wide and 137 ft. c. to c., inclined 19 deg. for the first 16 ft. and for the rest traveling horizonally over the mill storage bin. It is driven by a 10-hp., squirrel-cage, dust-proof motor, through a 21.3:1 reducer.

Distribution of the ore to the mill storage bin is effected automatically by a Link-Belt tripper operating on No. 8 conveyor. The tripper is driven by a 5-hp. motor, through a 40:1 reducer, spur gear and pinion, at 20 ft. per min., and has a plough attachment like that on the surge-bin tripper.

The mill storage bin is of steel construction, without internal cross-bracing. It is 37½ ft. deep, 25 ft. wide and 96 ft. long. The lower 15 ft. is tapered for the whole length of the bin to a width of 7½ ft. at the bottom. There are ten chutes, each 1½ ft. square, controlled by rack-and-pinion gates. The capacity of the bin is 4,200 tons, the live load approximately 3,000 tons.

A crew of eleven men—six on day shift, including a foreman, and five on night shift—operates the crushing plant during two 8-hr. shifts.

The crusher building is equipped with a Sly dust-collecting system, the filter being situated on top of the mill storage bin, into which the accumulated dust is discharged through five hoppers, the sides of which are sloped at 60 deg. The fan draws through a 20-in. pipe from the clean-air side of the filter under a 6-in. static head and is driven by a 20-hp., squirrel-cage, dust-proof motor, through a cog-belt drive at 725 r.p.m. Three 1-hp. motors, with speed reducers, operate the dust-envelope beaters and hopper rappers. The filter contains 260 34x38x1-in. envelopes covered with special cotton.

Suction lines draw the dust from above and below the cone crusher, as well as the rolls, surge bin, and all belt transfer points. The filter housing is insulated on the outside to reduce condensation within. Five tons of 82 per cent —325-mesh dust, assaying $2 per ton, are collected in two 8-hr. shifts.

As a result of this installation, regardless of the fact that the return air from the fan is discharged within the building, the atmosphere is practically free of dust.

Sixteen dust samples, taken with a circular konimeter, at the center of the fan discharge pipe, gave an average dust count of 230 particles per cubic centimeter—less than 5 microns.

Crushing costs, including labor, employees' insurance, supplies, maintenance, power and general expense, are given in Table IV.

Table IV—Crushing Costs

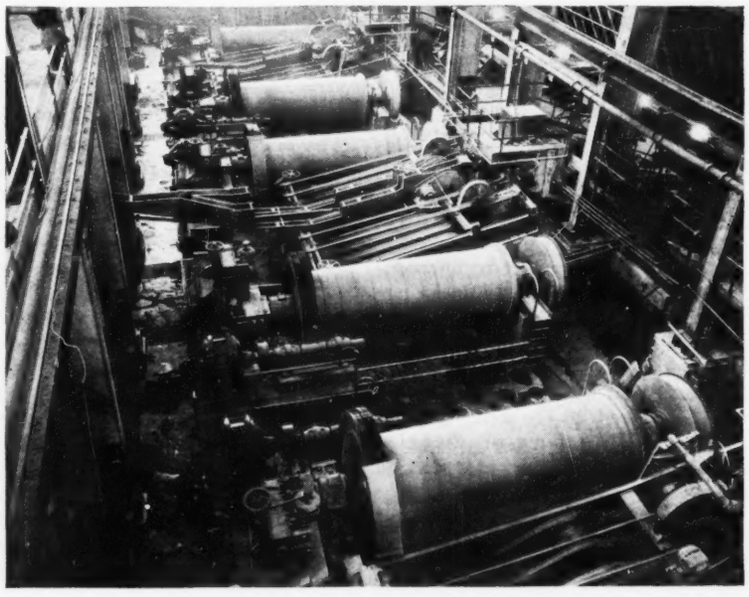
Symons cone crusher	$0.0221
Rolls	0.0386
Screens	0.0075
Conveyors	0.0207
Heating and lighting	0.0036
Supervision	0.0145
Total cost per ton	$0.1070

Flotation

1,200 Tons' Daily Capacity

FROM THE MILL STORAGE BIN the ore is fed through ten chutes to five horizontal belt feeders, each being 30 in. wide, with lengths varying from 4 to 20 ft. c. to c. They are driven individually by 2-hp., 550-volt, variable-speed motors, with a 550-volt resistance grid control, through 91:1 Falk reducers, giving a variable belt speed of 13 to 26 ft. per min.

The tube-mill floor. The five mills grinding to —65 mesh are seen



The belt feeders discharge to Nos. 9A, 9B, and 9C conveyors, operating at right angles. The first feeds Nos. 1 and 2 tube mills; the second Nos. 3 and 4, and 9C feeds No. 5. These conveyors are 24 in. wide by 19 ft. 3 in. c. to c., inclined at 18 deg. Each belt is driven by a 5-hp., squirrel-cage, dust-proof motor, through a 43.5:1 reducer, at 105 ft. per min. with a maximum capacity of 65 tons per hour.

Five Allis-Chalmers, 5x16-ft. tube mills grind the $\frac{3}{8}$-in. product from the crushing plant to minus 65 mesh at the rate of 2,100 tons daily. Each is driven by a 150-hp., 550-volt, 375-r.p.m. slip-ring motor, through a 13.0:1 double helical gear and pinion, giving a speed of 29 r.p.m. Each mill is loaded to the center with approximately 40,000 lb. of 2-in. forged-steel balls, and maintained at that level by adding 650 lb. daily. Moisture content of the pulp varies between 30 and 35 per cent. Wave-type white-iron liners, $2\frac{1}{2}$ in. thick, are used, with an approximate life of nine months.

Each mill is operated in closed circuit with a single No. 500 Denver Sub-A flotation cell and a Dorr classifier. The total pulp discharge is passed through 4-mesh screens, attached to the end of mills; the undersize goes to the cells, and the small amount of oversize, consisting of gangue and fine steel, passes directly to the classifiers. The tailings from the cells flow by gravity to the classifiers, the concentrate going to a receiving box. From this source a 2-in. Wilfley pump, driven at 1,100 r.p.m. from a lineshaft, pumps the material to the concentrate storage tank. The cells are operated at a dilution of 1:1, made up by the addition of water to the feed and through the cones.

The No. 500 cells have been equipped with cones operated on the hydraulic principle, making a combination machine (see Fig. 3 and photograph) for the purpose of removing gold which is too coarse to float, thus avoiding an accumulation in the tube-mill circuit. Every 24 hr. the tube-mill discharge is diverted to the classifiers. Water is then added for 15 min. to separate the gangue in the cells from the high-grade concentrate, after which a product consisting of sulphides and coarse gold is removed through a 4-in. plug valve equipped with a locking device. By this scheme 400 lb. of a rich material containing approximately $3,000 is recovered daily. This is immediately transferred to a tube mill in the cyanide circuit, with no evident increase in the value of the cyanide residue. Prior to the addition of the cones, the cells recovered 60 per cent of the total values, by flotation alone (as shown in Fig. 5 on the removal of gold from the grinding circuit). Since the introduction of the cones, extraction has increased to 75 per cent and a definite saving has been shown in the final flotation tailings.

Five 6x30-ft. duplex Dorr classifiers, driven by 5-hp. motors through V-belt drives, with the rakes operating at 18 strokes per minute on a $2\frac{1}{2}$-in. slope, return the sand by a 45-deg. launder to the tube-mill drum feeders. The overflow at 2:1 dilution gravitates to a surge box.

The overflow from the classifiers is elevated by a 6-in. Morris sand pump from the surge box to a distributing

Table V — Screen Analyses of Tube-Mill and Classifier Products Flotation Section

Mesh	Tube-Mill Feed (Dry)	Tube-Mill Discharge	Classifier Rake Return	Classifier Overflow
Plus 3	....	....	....	....
Plus 4	2.2	....	....	....
Plus 6	14.6	0.1	....	....
Plus 8	15.7	0.2	0.2	....
Plus 10	15.2	0.6	0.1	....
Plus 14	8.6	0.8	0.2	....
Plus 20	8.1	1.4	0.9	....
Plus 28	6.6	3.1	5.3	....
Plus 35	5.5	7.4	12.8	....
Plus 48	4.3	10.9	24.0	....
Plus 65	3.1	12.4	31.1	6.2
Plus 100	3.0	12.9	14.7	11.3
Plus 150	1.9	8.6	5.1	10.7
Plus 200	1.7	6.5	2.3	14.5
Minus 200	9.5	35.1	3.3	57.3
	100.0	100.0	100.0	100.0

Table VI — Screen Analyses and Assays of Primary and Secondary Cell Tailings

PRIMARY CELLS

Mesh	Percentage Weight	Cumulative Per Cent	Assay	Contents, Cents
Plus 65	6.5	6.5	$0.95	6.2
Plus 100	12.1	18.6	0.50	6.0
Plus 150	10.5	29.1	0.36	3.8
Plus 200	12.3	41.4	0.40	4.9
Minus 200	58.6	100.0	0.33	19.3
	100.0	100.0		40.2

SECONDARY CELLS

Mesh	Percentage Weight	Cumulative Per Cent	Assay	Contents, Cents
Plus 65	6.8	6.8	$0.41	2.8
Plus 100	12.4	19.2	0.30	3.7
Plus 150	10.7	29.9	0.25	2.7
Plus 200	12.2	42.1	0.29	3.5
Minus 200	57.9	100.0	0.21	12.1
	100.0	100.0		24.8

Table VII—Chemical Analysis of Flotation Tailings

AVERAGE FOR SIX MONTHS

	Per Cent
Iron pyrite	0.10
Silica	58.28
Combined water and carbon dioxide	9.10
Ferric oxide	6.79
Aluminum oxide	15.07
Calcium oxide	5.61
Magnesium oxide	2.82
Sodium and potassium oxide	2.95
Gold, per ton	24c
Specific gravity of dry tailings	2.72

Table VIII — Screen Analyses and Assays of Flotation Concentrates to Cyanidation

Mesh	Percentage Weight	Assay Per Ton	Contents Per Ton
Plus 65	2.5	$90.20	$2.26
Plus 100	7.8	74.08	5.78
Plus 150	10.8	70.36	7.60
Plus 200	11.9	68.70	8.17
Minus 200	67.0	54.57	36.56
	100.0		$60.37

Table IX — A Chemical Analysis of Flotation Concentrate

AVERAGE FOR SIX MONTHS

	Per Cent
Iron pyrite	64.89
Silica	15.41
Ferric oxide	3.17
Aluminum oxide	8.61
Calcium oxide	3.19
Magnesium oxide	2.50
Gold, per ton	$62.21
Silver, ounces per ton	0.01
Copper (as chalcopyrite)	0.61
Zinc (as sphalerite)	0.71
Lead (as galena)	0.02
Specific gravity of dry concentrate	3.80 to 4.20

An aisle on the flotation floor

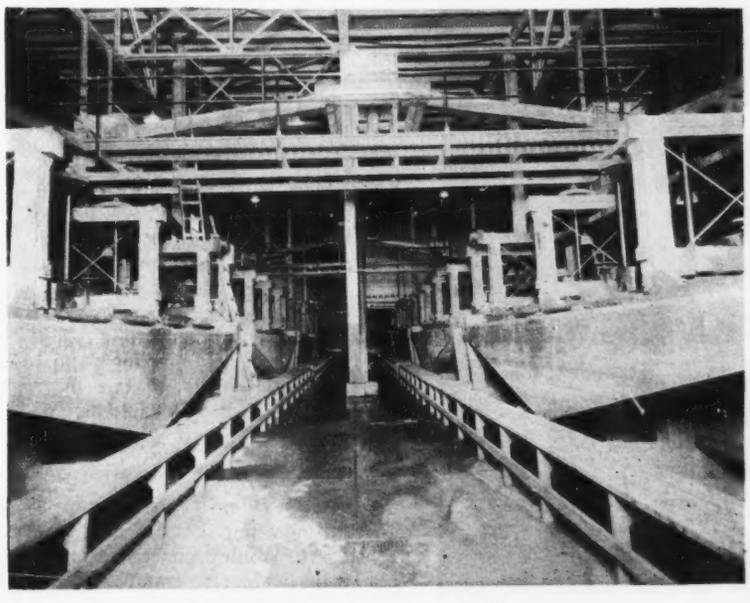


Table X—Flotation Costs

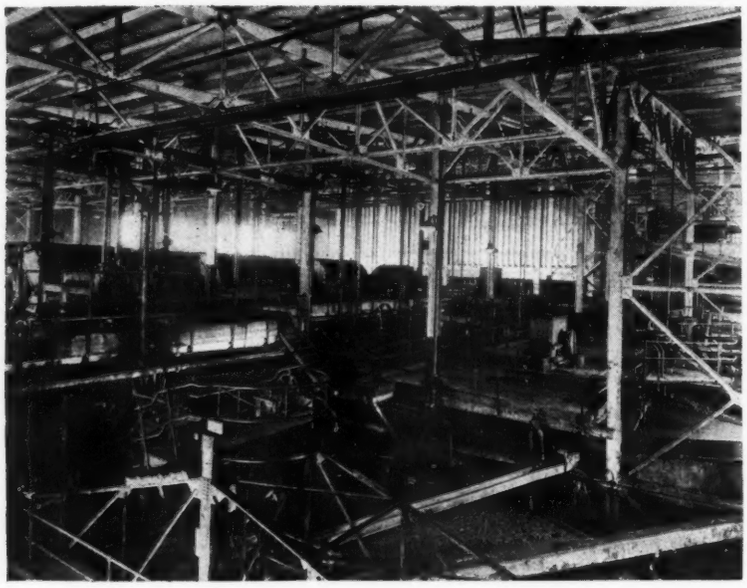
Tube milling	$0.1294
Classification	0.0100
Flotation machine	0.0616
Reagents	0.0772
Pumping and elevating	0.0254
Filtration	0.0170
Tailings disposal	0.0217
Heating and lighting	0.0037
Supervision	0.0173
Total	$0.3633

system, feeding the primary flotation cells. This is equipped with white-iron liners and a nickel-steel impeller, and is driven at 810 r.p.m., against a 42-ft. head, by a 50-hp. motor.

From the distributing system, the pulp flows by launders to 48 No. 24 Denver Sub-A primary cells, arranged in eight units of six cells each, one-eighth of the feed going to each unit. The impeller in each cell is driven by a 5-hp., squirrel-cage motor, through a V-belt drive at 265 r.p.m.

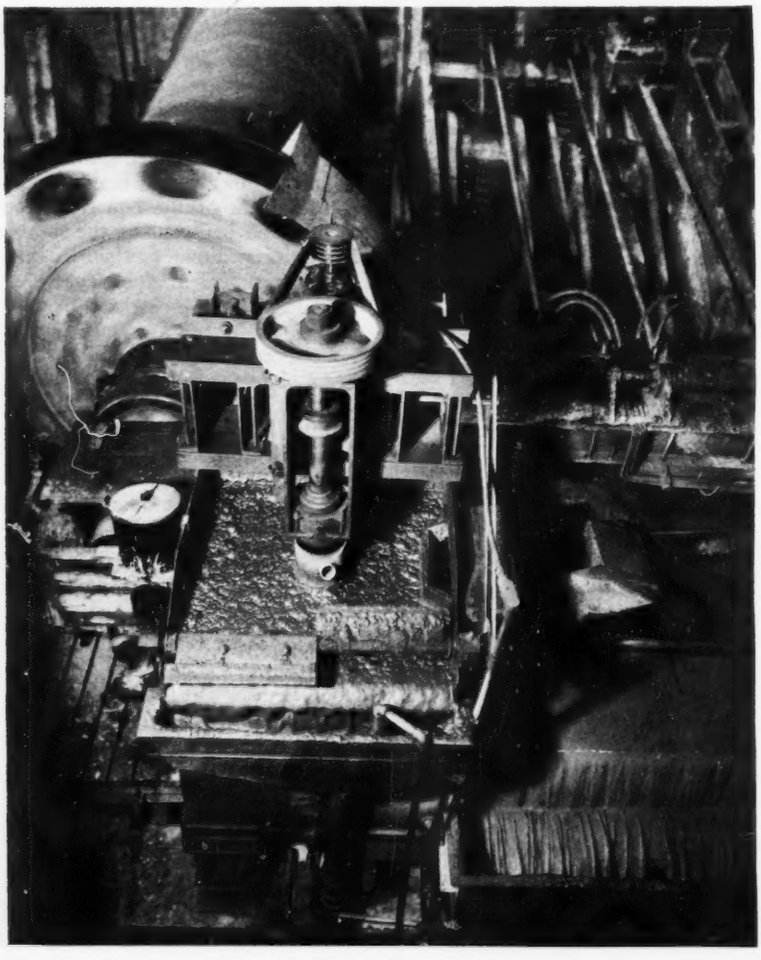
Primary cell tailings flow to a pump box, from which they are sent to the secondary-cell distributing system, through a 6-in. Morris sand pump, driven at 864 r.p.m. by a 50-hp. motor, with a V-belt drive, against a 47-ft. head. The pulp goes by gravity to 36 secondary cells. Save for the number of cells, this circuit is a duplicate of the primary.

The tailings from the secondary circuit are pumped to waste through an 8-in. horizontal pipe line, 1,000 ft. long, by a 6-in. sand pump, driven at 912 r.p.m. by a 50-hp. motor.



A view of the filter floor (in the background)

Combination flotation cell and hydraulic cone, operating in each tube-mill-classifier circuit. It receives the tube-mill discharge. See Fig. 3



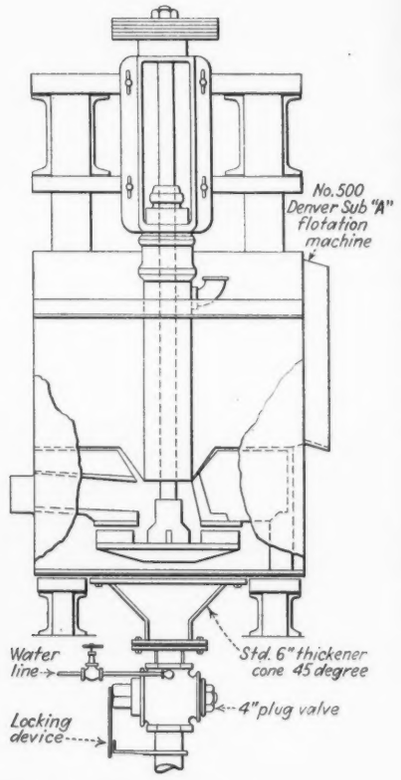


Fig. 3—Sketch, in elevation, of combination flotation-cell hydraulic-cone unit shown at the lower left

The work done by flotation is shown in the screen analyses (Table VI) of the primary and secondary cell tailings. These analyses show the distribution of the values in the different sizes, and their response to flotation treatment.

From the chemical analysis (Tables I and VII) of the feed and tailings in the flotation circuit one may note that the pyrite is amenable to this process, as a recovery of 98.78 per cent is indicated.

The Pilot Table—About 50 tons of tailings per 24 hr. are pumped to a 15-ft. Wilfley concentrating table, by a 2-in. Morris sand pump, directly

coupled to a 5-hp. motor, operating at 750 r.p.m. The table is driven at 260 strokes per mintue by a 5-hp. motor, with a V-belt drive, and acts as a pilot to guide the flotation operators.

Concentrates from primary and secondary cells pass to the concentrate storage tank, 12 ft. in diameter by 11 ft. deep, equipped with a Dorr agitator mechanism, driven at 10 r.p.m. from the same lineshaft that drives the 2-in. Wilfley pump on the tube-mills cell concentrate circuit. This shaft is driven by a belted 10-hp. motor. The pulp is thickened in this tank to 1.7:1 dilution by twelve 6-ft. (deep) by 3½ ft. clarifier leaves, under a constant vacuum of 24 in. before it is fed to the dewatering filters. Leaves are cleaned daily.

The concentrate is drawn through a 2-ft. discharge cone at the tank bottom, by a 4-in. Wilfley pump operating at 800 r.p.m., and driven by a 20-hp. motor through a V-belt drive, then discharged to three American dewatering filters, against a 35-ft. head, the overflow returning to the tank. The filters are 8 ft. 6 in. in diameter, with eight disks, and are partitioned in the pulp tank, by steel plates welded in, each compartment being fed from above, through 1-in. nipples from the main feed line. This prevents coarse concentrate from building up and sticking the filters.

The cake is washed with water to remove soluble flotation reagents. The filter bags are of 32-oz. twill, and one bag from each disk is removed every 24 hr. for washing. Each filter is driven, at variable speeds, by a 5-hp. motor. A vacuum of 24 in. is maintained. The filtrate enters a 3-ft. (diameter) by 8-ft. steel vacuum receiver, from which it is pumped by a 4-in. Rees Roturbo centrifugal suction pump, to a hemispherically bottomed, steel storage tank, 16x24 ft. deep and having a capacity of 36,000 Imp. gal.

The cake from the filters, discharged at 20 per cent moisture, drops to individual repulpers, below each filter. These are semi-cylindrical steel tanks, 13 ft. by 2 ft. 4 in. wide, equipped with paddle shafts, to which are attached 24 straight steel blades, 12 in. long by 3 in. wide by ½ in. thick. The repulper shafts are driven at 100 r.p.m. by a belt drive from a line shaft, and the cake is repulped to 50 per cent moisture with barren cyanide solution.

Flotation Reagents — The American Cyanamid Company's promoter reagent is fed into the tube-mill circuit at the rate of 0.18 lb. per ton, by a Denver Equipment Company's reagent feeder. No. 25 Aerofloat is fed at the rate of 0.08 lb. per ton to the pulp entering the primary and secondary circuits. It is also fed to the tube-mill circuit cells, and in case of the necessity of any additional frother, drip pails are used.

Flotation costs, including labor, employee insurance, maintenance, power, and general expense, are given in Table X on page 477.

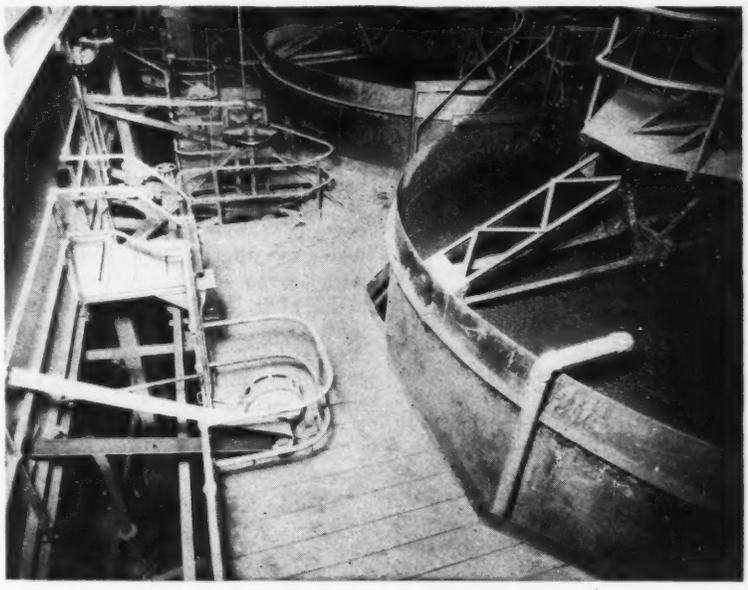
Cyanidation

Capacity 400 Tons Per Day

APPROXIMATELY 230 tons of concentrate per day flows from the dewatering filters to the drum feeder of No. 6 Allis-Chalmers 5x16-ft. tube mill, which is in closed circuit with a 6x30-ft. duplex Dorr classifier. The mill is driven at 29 r.p.m. by a 150-hp., 550-volt, 375 r.p.m., slip-ring motor through a 13.0:1 double helical gear and pinion. It is loaded with 30,000 lb. of 2-in. forged steel balls, 800 lb. being added daily. Moisture content of the pulp is maintained at 40 per cent. Rubber liners, 1 in. thick and having a life of approximately two years, are attached to the mill shell by manganese-steel retaining bars spaced 15 in. c. to c. End liners are of white iron, 3 in. thick, and are replaced every sixteen months.

The classifier is operated at a slope of 2⅛ in. to the foot, and is driven by a 5-hp. motor through a V-belt and a Reeves variable-speed drive, giving a range of ten to twenty strokes per minute on the rakes. The overflow product at 7:1 dilution shows 92 per cent minus 200 mesh, and 80 per cent minus 325, and passes to a pump box, where it joins the overflow from No. 7 classifier at 1:1 dilution, thus making an average dilution of 3:1 in the agitators. The pulp is then pumped by a 6-in. Morris sand pump to the first of six agitators, which are in series. The pump is driven at 810 r.p.m. by a 50-hp. motor, through a V-belt, against a discharge head of 23 ft. For aëration purposes, a portion of the pulp is drawn from the bottom of No. 1 agitator, by a bypass arrangement, and joins the current flow.

The six Dorr agitators, 24 ft. in diameter by 20 ft. deep, are driven at 13 r.p.m. by individual 5-hp. motors, through 4.22:1 Falk reducers and belt drives. Each is equipped with a 36-in. discharge cone and a 6-in. air lift, which circulates the pulp back to the tank, or sends it forward to the next agitator, if so desired. Each tank is connected to the one following by 6-in. pipes, located 3 ft. from the top, as well as welded steel launders stepped down from tank to tank, to take care of current froth.



On the cyanidation floor. A pipe-guarded motor above one of the agitators

The pulp is passed from the final agitator to a Dorr bowl classifier by a 6-in. Morris sand pump, driven at 920 r.p.m. by a 50-hp. motor through a V-belt, against a 53-ft. head.

The bowl classifier is 20 ft. in diameter with a reciprocating rake compartment, 6x30 ft. long, having a slope of 2 in. to 1 ft. and raking at seven strokes per minute. The bowl section, which slopes at 2 in. to 1 ft., has a rake speed of 1⅜ r.p.m. The classifier is driven by a 10-hp. squirrel-cage motor (mounted on the tank compartment) through a V-belt drive to a line shaft, which in turn operates both the reciprocating and bowl rakes, through belt drives.

The reciprocating rakes will not handle the return product, under the conditions necessary to obtain the desired fineness in the overflow; therefore

the bowl is operated on the principle of a settling cone, the underflow discharged by gravity through a spigot at the bottom of the tank to the feed end of No. 7 regrinding tube mill, with a moisture content of 40 per cent. The bowl overflow travels to the thickener through launders at a dilution of 8-10 to 1, giving a product of 98 per cent minus 325 mesh.

Underflow from the bowl classifier is ground in No. 7 tube mill, operating in closed circuit with a duplex classifier. In mechanical details, the mill and classifier are identical with No. 6, except that 1¼-in. balls are used, added at the rate of 800 lb. daily. Classifier overflow from this unit is kept at 1:1 dilution and joins the flow from No. 6.

Bowl overflow is split to four points on the periphery of a 50-ft. Dorr tray thickener, 14 ft. deep, driven by a 5-hp. motor, at one revolution in 5 min. through a 6.93:1 Falk reducer and belt drive.

Froth is taken care of in the tray compartment, also a large percentage of the solids, and the partially clarified solution flows down the center well to the bottom section, from which all the solution is drawn, at two points on opposite sides of the thickener. Pulp is removed at 50 per cent moisture by two 4-in. Dorr duplex diaphragm pumps, two of which are driven from a line shaft through a belt and a Reeves variable drive. The other two are driven by a direct belt drive from the same line shaft. The suction of one pump is submerged in the froth on top of the thickener, while two others pump the pulp from the bottom of the tray, the remaining one being used to take care of the pulp from the bottom of the tank (Fig. 4).

Thickened pulp flows from the diaphragm pumps to a surge tank, 6 ft. in diameter by 12 ft. deep, from which it is sent to the first stage filters by a 4-in. Wilfley pump, driven at 890 r.p.m. by a 15-hp. motor, through a V-belt. Overflow from these filters returns to the tank.

Three stages of filtering are required for elimination of pregnant solution from the cake. For this purpose there are five American filters—two on both first and second stages and one on the third stage. Each machine has eight 8 ft. 6 in. disks, with drives like those used for dewatering concentrate. Every shift, two rows of bags are removed from one filter in each stage and acid-treated. This completes the cycle in eight days.

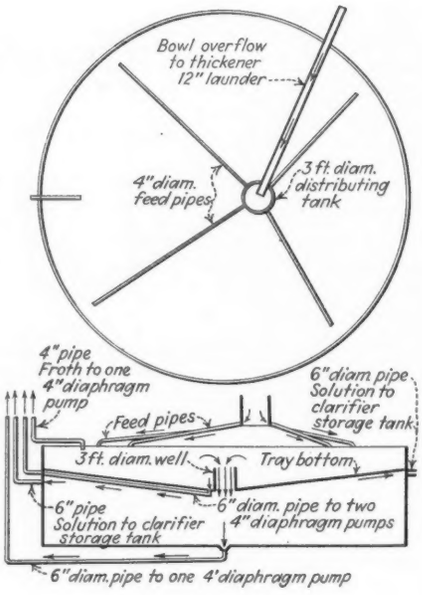

Bowl overflow to thickener 12" launder
4" diam. feed pipes
3 ft. diam. distributing tank
4" pipe Froth to one 4" diaphragm pump
6" diam. pipe Solution to clarifier storage tank
Feed pipes
3 ft. diam. well
Tray bottom
6" pipe Solution to clarifier storage tank
6" diam. pipe to two 4" diaphragm pumps
6" diam. pipe to one 4' diaphragm pump

Fig. 4—Feed and discharge arrangement for 50-ft. tray thickener

From the first stage, the cake, after being sprayed with barren solution, is discharged at 20 per cent moisture to re-pulpers like those used on the dewatering filters. The pulp is diluted with barren solution to 1:1 and goes to a 12x14-ft. steel surge tank, equipped with Dorr agitator mechanism and driven from a line shaft at 10 r.p.m. It is then pumped against a 25-ft. head to the second stage by a 4-in. Wilfley, similar to that used on the first stage.

Second- and third-stage filtering is identical with the first, except that one machine is used on the final stage. Periods occur when carbonaceous material is present in the ore and is passed to the cyanide circuit. This is indicated by the abnormal assay of froth on the agitators and thickener, resulting in an increase in the residue value. This condition is overcome by diverting the washed residue to two six-cell Denver Sub-A flotation units, before discharging it to waste. Due to the effect of the cyanide in the prior treatment, approximately 95 per cent of the material is depressed, and the values are recovered in the form of a concentrate and returned to the cyanide circuit for further dissolution.

The filtrate from the first-stage filters is drawn into a 3x8-ft. steel receiver, from which it is pumped to the thickener against a 20-ft. head by a 4-in. Rees Roturbo (like that used on the dewatering filtrate) showing a vacuum of 26 in. at the intake. The combined filtrate from the second and third stages is drawn into a duplicate receiver, from which it is sent to a 4½x10-ft. steel storage tank by a 6-in. Rees Roturbo pump, operating against an 8-ft. head and showing 26 in. of vacuum on the suction line. This is driven at 1,500 r.p.m. through V-belts by a 25-hp. motor. From the tank, the filtrate and excess thickener solution is pumped to the mill cyanide storage tank, by a 7x9-in. Aldrich triplex pump. The storage tank is of the same size and located at the same elevation as that receiving the dewatering filtrate. The Aldrich pump is operated against a 69-ft. head by a 20-hp. motor through a V-belt drive and 5:1 gear and pinion, at 76 strokes per minute.

Pregnant solution from the thickener goes to a 15x26-ft. steel tank; from here a 7x9-in. Aldrich triplex pump sends it to two clarifying presses, 42x42 in. with 25 frames each, using 10-oz. canvas. The sluicing mechanism is driven from a line shaft through a worm gear and belt at 3 r.p.m. Canvases are cleaned and acid-treated approximately every five days, and have a life of three weeks.

Clarified solution flows to a 15x26-ft. steel storage tank, from which it is pumped by a 4-in. Rees Roturbo, direct-coupled to a 15-hp. 1,500-r.p.m. motor, to the top of a 5x12-ft. Crowe vacuum receiver, situated 44 ft. above the bottom of the pregnant storage tank. Between

Table XII—Analysis of Barren Solution

	Per Cent
Total solids	1.58
Total cyanogen	0.36
Free cyanide	0.17
Alkali (before removing CO_2)	0.19
Alkali (after removing CO_2)	0.227
Lime	0.033
Ferrocyanide	0.0054
Thiocyanate	0.210
Thiosulphate	0.410
Chlorine	0.0086
Sodium and potassium oxide	0.498
Magnesium oxide	0.0035
Copper	0.090
Lead	Nil
Gold	Trace
Silver	Trace
Aluminum (Al_2O_3)	0.0036
Arsenic	Nil
Antimony	Nil
Zinc	0.041

Table XI—Analysis of Raw Precipitate

	Per Cent
Gold	33.29
Silver	5.76
Copper	11.41
Lead	19.41
Zinc	14.43
Iron	1.46
Sulphur	6.88
Silica	1.48
Aluminum oxide	1.55
Calcium oxide	1.90
Magnesium oxide	0.62
Sodium oxide	1.31
Total	99.50

Table XIII—Operating Costs, Cyanidation

Tube milling	$0.0327
Agitation	0.0105
Classifying	0.0067
Thickening	0.0052
Pumping and elevating	0.0292
Filtration	0.0323
Clarification	0.0073
Precipitation	0.0157
Reagents	0.1127
Heating and lighting	0.0037
Supervision	0.0174
Mill alterations	0.0041
Total	$0.2775

Table XIV — Consumption of Grinding Media and Chemicals

April 1-July 31

	Pounds Per Ton
Grinding media, 2 in. (flotation section)	1.791
Grinding media, 2 and 1½ in. (cyanide section)	0.585
Total	2.376
Cyanide	0.628
Lime	1.106
Zinc dust	0.084
Lead acetate	0.022

this tank and the Crowe receiver a 5x3¾-in. Venturi tube measures and records the solution tonnage precipitated.

In the Crowe system a Rix-Gardener dry vacuum pump is used and operates from the line shaft which drives the sluicing mechanism on the clarifying filters. The shaft is driven by a 10-hp. motor through a belt drive.

A 7x9-in. Aldrich triplex pump sends the solution, to which lead acetate is added, from the Crowe tank to the precipitation presses. Zinc dust is fed by a Merrill feeder to the intake of the pump, through a 2 ft. 4 in. (diameter) by 2 ft. 8 in. cone. Strong cyanide solution is also added at this point to aid precipitation.

Precipitation is effected at the rate of 1,600 tons per day in three 52-in. triangular Merrill presses, each with 22 frames. Two presses are used continuously, the third acting as a spare. Twelve-ounce canvas and unbleached sheeting are used. At intervals, the canvases are acid-treated. They last about three months. The sheeting is burned and mixed with the precipitate at each clean-up. Every ten days, the presses are cleaned, the resultant precipitate being transferred to an acid-treating tank in the refinery through a 10-in. pipe in the press-room floor.

From the presses the barren solution flows to a 12x14-ft. steel storage tank, from which it is pumped, by a 7x9-in. Aldrich, to the filter spray system and to supply gland solution on pumps. Surplus solution goes to the mill cyanide storage tank. A pressure valve arrangement on the pump-discharge line maintains a pressure of 50 lb. per square inch on the filter sprays and glands. Analysis of the barren solution is given in Table XII.

Vacuum for filtration is maintained at 28 in. by three 23x12-in. Ingersoll-Rand vacuum pumps, having a capacity of 2,188 cu.ft. per min. each; two are in continuous service and one is a spare. They are driven by 100-hp. motors, through short belt drives equipped with idlers and operated at 190 r.p.m.

Air required for agitator air lifts and filtration is supplied at 20-lb. pressure by a 2,000-cu.ft. Size 11, Type 23A, Sentinel Alley & McClellan air compressor. This is driven at 290 r.p.m. through a V-belt drive by a 200-hp. motor, and is equipped with the usual lubricating system, intake air filter, and receiver. One spare Sullivan air compressor of 1,000-cu.ft. capacity is available. This is driven at 142 r.p.m. through a flat belt drive by a 125-hp. motor.

A crew of 45 men, composed of operators, mechanics, samplers, and roustabouts, is employed in the flotation and cyanide sections.

Operating costs for cyanidation, based on original tonnage, including labor, employee's insurance, supplies, maintenance, power, and general expense, are given in Table XIII.

Consumption of grinding media and chemicals, based on total tonnage treated from April 1 to July 31 inclusive, is indicated in Table XIV.

Cassel sodium cyanide is used and is added at Nos. 6 and 7 tube mills and agitation tanks, as well as at the zinc cone. The lime used has a solubility of approximately 85 per cent; it is slaked and added periodically to the tube-mill feed by the operators. Cyanide and lime strength in the solutions are maintained at 3.5 lb. and 0.75 lb. respectively.

Tailings Disposal (Entire Mill)—All mill tailings are disposed of at 33 per cent solids, through an 8-in. wood-stave, self-draining line, against a head of 20 ft. The line discharges at a point 1,000 ft. north of the mill, where the elevation permits of gravity flow to a natural tailings pond, protected at low points by sand and gravel embankments, built to a height of 20 ft. The area of the pond is 125 acres, with a storage capacity of approximately three million tons. The solution overflow is controlled by weirs located at a central point of the lower dam, and flows to Porcupine Lake, 5 miles away.

Mill Control

FRESH WATER for dewatering filter sprays and drinking fountains is piped from the Mattagami River, at 90-lb. pressure. With this exception, all the water used in the mill is obtained from Pearl Lake, 800 ft. away. It is pumped directly to a 16x24 ft. deep, hemispherically bottomed storage tank, in the crusher house section, 63 ft. above the tube-mill floor, the tank overflow being returned through a 6-in. pipe. A typical analysis of the water over a period of several months is given in Table XV. The impurities, indicated by this analysis, have no bad effect on extraction in the flotation circuit.

Table XV—Analysis of Mill Water Supply

	Per Cent
Total solids	0.0473
Silica	0.0008
Organic matter	0.0078
Iron and aluminum oxide	0.0004
Calcium oxide	0.0098
Magnesium oxide	0.0033
Sodium and potassium oxide	0.0054
Total sulphates (as SO_4)	0.0209
Alkalinity	0.0013
Dissolved oxygen, milligrams	0.0054

Table XVI—Average Temperatures at Mill

	January and February Deg. F.	July and August Deg. F.
Outside air	8	63
Inside air	59	70
Water	36	66
Flotation solutions	48	73
Cyanide solutions	64	79

Table XVII—Alkalinity of Flotation Solutions

	pH
Lake water (Pearl Lake)	7.80
Tube mill feed	8.10
Tube mill discharge	8.90
Classifier overflow	8.70
Primary cell feed	8.75
Primary cell discharge	8.60
Secondary cell feed	8.80
Secondary cell discharge	8.40
Filtrate from concentrate	8.30

Table XVIII—Metallurgical Results, April 1-July 31, 1933

Tons milled	249,100
Head assay	$7.61
Tons of flotation concentrate	27,727
Ratio of concentration	8.98:1
Recovery by flotation, per cent	97.06
Concentrate assay	$66.36
Flotation tailing assay	0.253
Cyanide residue assay	0.849
Combined residue assay	0.319
Final recovery, per cent	95.80

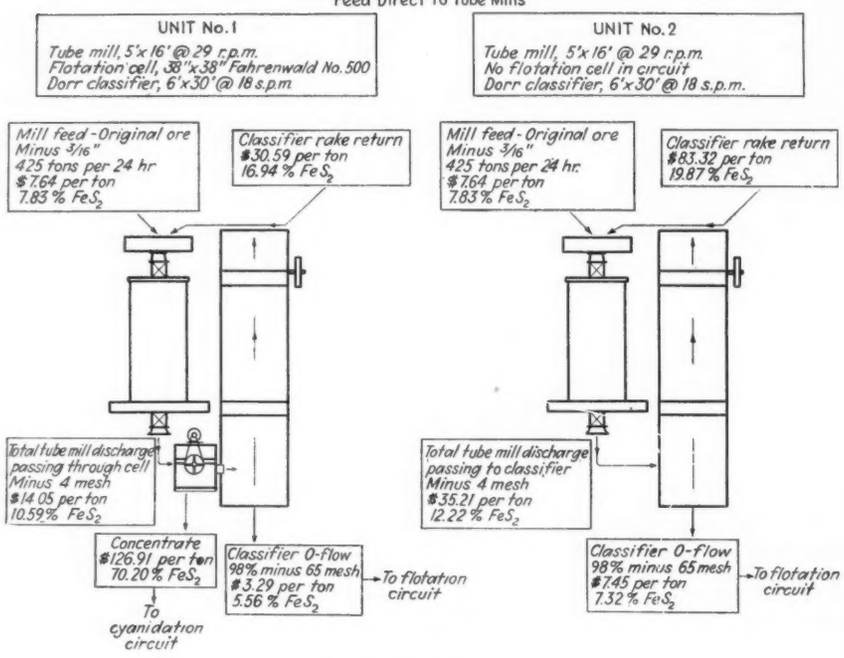
The temperature of the fresh water and the mill solutions varies considerably between the summer and winter months, with no appreciable difference being noticed in the metallurgical results. Table XVI gives the average temperatures for these periods.

Alkalinity determinations are made periodically (with a Leeds and Northrup *pH* indicator) on the water supply and the different solutions in the flotation circuit. Averages are presented in Table XVII. The slight variations of the *pH* readings in the flotation circuit are due to the fact that no other compounds are used, aside from the regular collecting and frothing reagents.

Dilutions of the various products in the circuit are controlled and checked hourly by the operators. On the flotation products, with the exception of the concentrate, determinations can be made by specific gravity methods, but in the cyanide section, owing to the variation in the specific gravity of the dry concentrate, pulp density indicators and specific gravity methods are unreliable and dilutions must be determined by the procedure of weighing and drying the pulp.

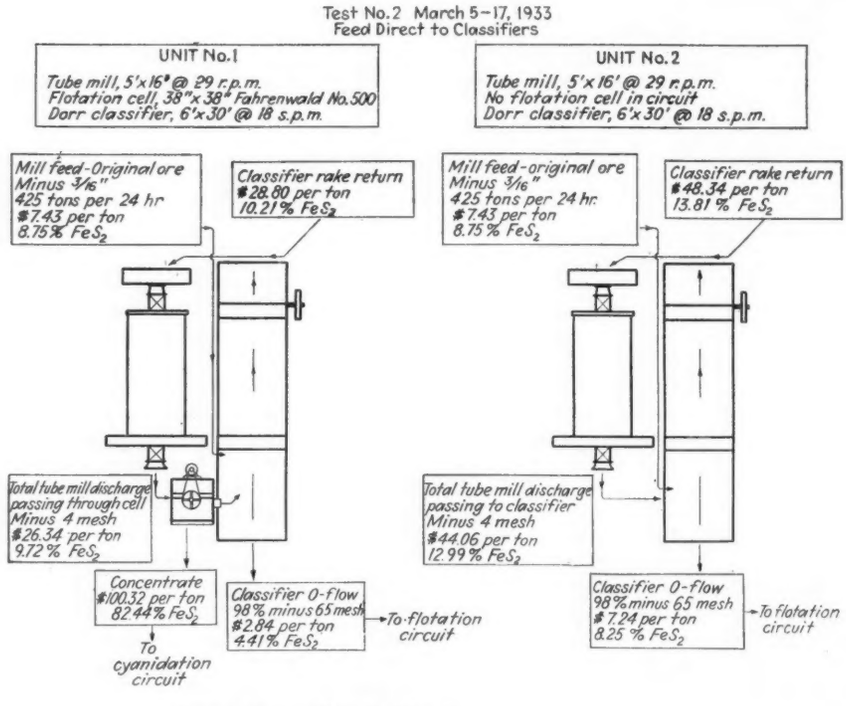
A spacious, well-lighted laboratory with the necessary apparatus is available for conducting investigations as well as checking the various steps in plant treatment. It is equipped with experimental agitators, flotation machines, filters and dryers, in addition to a Wilfley table, a Tyler Ro-Tap screening machine, and the usual chemical and auxiliary equipment.

Mill heads are sampled every half hour on the ¾-in. product feeding the tube mills. Approximately 200 lb. is taken each shift and reduced to 50 lb. by means of a Jones riffle. The sample is then sent to the assay office, where



Extraction—Circuit No. I

Product	Tons	Assays	Content	Per Cent Recovery	Ratio of Concen	FeS_2 Tons	FeS_2 %
Conc.	14.93	126.91	1897.87	58.45	28.46:1	10.48	31.5
Tails	410.07	3.29	1349.13	41.55	.....	22.80	68.5
Total	425.00	7.64	3247.00	100.00	.....	33.28	100.00



Extraction—Circuit No. I

Product	Tons	Assays	Content	Per Cent Recovery	Ratio of Concen.	FeS_2 Tons	FeS_2 %
Conc.	20.00	100.32	2006.40	63.54	21.25:1	19.40	52.22
Tails	405.00	2.84	1151.35	36.46	.....	17.75	47.78
Total	425.00	7.43	3157.75	100.00	.....	37.15	100.00

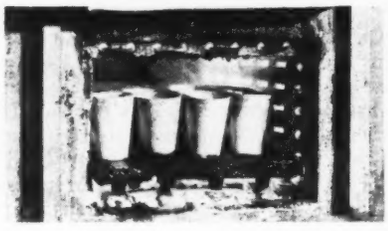
Fig. 5—RESULTS OF COMPARATIVE TESTS showing gold values removed from grinding circuit by flotation machine (Fig. 3).
Samples taken at intervals of 15 min. on day shift for a period of eleven days

it is ground and assayed for its contents.

Flotation concentrate, flotation tailings, and cyanide residue are sampled by Geary-Jennings automatic machines. Each shift, about 100 lb. of the flotation tailings and 50 lb. of concentrate and cyanide residue are taken. These samples are filtered in Denver pressure filters, dried, and sent to the assay office.

Hand samples are taken periodically by the operators from the classifier overflow and primary tailings in the flotation section, together with the classifiers, agitators, thickeners, and filters in the cyanide circuit. A special sample of the cyanide residue from the third-stage filter is taken and divided into two parts. One part is dried to determine the moisture content; the other is filtered and the resultant solution assayed. From these results the soluble gold values are calculated.

Solutions in the cyanide circuit are sampled continuously by drip wires, cutting through the flow. These samples are titrated for cyanide and lime, and then assayed.



All assays are reported in dollars and cents, based on a price of $20.6718 per troy ounce for gold.

Owing to the variation of values in the mill products, different quantities are taken for assay. A 3 assay-ton charge is used for the mill feed, whereas 10 assay-tons are required on flotation tailings. Flotation concentrate and cyanide pulp samples are roasted, ½ to 3 assay-ton charges being used.

All cyanide solution values are determined by evaporation in lead boats, at a temperature below boiling. Duplicate assays of the pregnant and barren solutions are made, taking from 3 to 10 assay-tons.

Bullion determinations are made by standard methods. Three-hundred-milligram lots are weighed in triplicate; two for gold, and the other for gold and silver. Other refinery products, such as matte and slag, are assayed by the fire or wet method.

Although head samples are taken daily, they are only indicative of the production that may be expected. Actual extraction is based on bullion produced and the assay value of the automatically sampled tailings. As a general rule, the value of the bullion recovered will be 1 to 2 per cent higher than that called for by head assay.

Metallurgical results, for the period from April 1 to July 31, are shown in Table XVIII.

In the Refinery

GENERAL PROCEDURE in the refinery is as follows: The precipitate is removed from presses, acid-treated, weighed, fluxed, melted, and refined, producing a marketable bullion. Forty-eight hours are required for complete treatment.

The precipitation presses in the mill are cleaned three times per month. At clean-up time, the spare press is started, the feed to the two full presses is stopped, and air blown through them for half an hour to dry the precipitate. They are then opened and the precipitate scraped into trays, 43x64x12 in. deep. From the trays a sample is taken and immediately sent to the assay office for a copper analysis. The rest is shoveled into a hopper which discharges directly into the acid-treating tank in the refinery, for removal of zinc and copper. Approximately 2,000 lb. of raw precipitate, assaying about $80 per pound, are obtained from each clean-up.

Table XIX—Bullion Analysis

	Per Cent
Gold	84.10
Silver	11.96
Copper	3.34
Lead	0.48
Zinc	0.02
Iron	0.08
Total	99.98

The tank is of wooden construction, lead-lined, and 10 ft. in diameter by 6 ft. deep. The agitator paddles are also of wood and operate at 22 r.p.m. The tank is completely covered in and is equipped with a No. 6 Buffalo Forge ventilating fan which operates at 1,250 r.p.m., drawing the fumes from the tank and discharging through the refinery roof. The agitator mechanism and fan are driven from a common lineshaft by a 5-hp., 550-volt, 800-r.p.m. motor. The motor is equipped with a ventilating arrangement which draws outside air through the commutator, thus protecting it from the action of the fumes.

Zinc and Copper Treatment

Zinc Treatment—The tank is partly filled with water before the precipitate is added, so that a pulp of approximately 3:1 dilution is obtained. Two or three carboys of sulphuric acid are added, and the charge is agitated for 2 or 3 hr. Steam is admitted to the tank during both zinc and copper treatments. The tank is then filled with water, allowed to settle, and the foul solution siphoned and run through the filter press. Two or three carboys more of acid are then added, and agitation is continued for 5 or 6 hr., after which the tank is again filled with water, allowed to settle, and the clear solution siphoned. This second wash is to ensure that the bulk of the foul solution has been removed; otherwise trouble would be encountered in the copper treatment.

Copper Treatment — A sample is taken from the tank and tested with sulphuric acid and manganese dioxide to determine the amount of these chemicals required to attack the copper. The chemicals are then added in small quantities until a good copper color is obtained. Agitation is continued for several hours until colorimetric tests indicate that no more copper is being dissolved. Should the precipitate contain a large amount of copper, the tank is filled with water, decanted, and the manganese dioxide treatment repeated as often as is necessary to reduce the copper content to about 4 per cent, which will give a bullion over 950 in total fineness.

When the acid treatment is finished, the tank is filled with warm water, and the pulp is run by gravity into a 3½ ft. (diameter) by 3 ft. cone-shaped monteju, made of ¼-in. steel plate. When it is filled, the inlet from the tank is closed and air admitted to the monteju. This forces the pulp, under a pressure of 20 lb. per square inch, into a 30-in. Perrin press with 21 frames, paper being used as the filter medium. The process is repeated until the press is full. Tailing solution from the press flows to a sump, from which it is pumped to a sand filter by a 1½-in. acid-resisting pump, directly driven at 1,420 r.p.m. by a 1-hp., 550-volt motor. The sand filter is 9½ ft. in diameter and 8 ft. deep, and is used to save any precipitate which may leak through the press. Solution from the filter goes to waste.

When the press is opened, the precipitate is scraped into 5x7 ft. by 8 in. trays, made of ⅛-in. steel plate. Here again it is sampled, after which the press is refilled and the process repeated until the acid-treating tank is emptied and sluiced out. The samples from all trays are combined and the moisture content determined, the precipitate being shoveled into tubs, weighed, and the dry weight calculated. The average weight of the material after acid-treating is approximately 1,400 lb., assaying about $115 per pound. It is then returned to the trays and thoroughly mixed with a flux consisting of 10 per cent soda ash, 10 per cent borax glass, 2½ per cent fluorspar, and 2½ per cent silica sand.

The mixture is charged into two No. 2 Rockwell, oil-fired, tilting furnaces, previously heated for about 1 hr. Each furnace is lined with 1,400 lb. of "Carbofrax," moistened with 7 per cent of water, tamped into place and fired. Furnace linings last about three months. Air is used to force fuel oil from the tanks to the furnaces, under a pressure of 50 lb. per square inch. A No. 4 Roots blower supplies air for combustion at 1½-lb. pressure per square inch, and is operated at 220 r.p.m., with a capacity of 1,400 cu.ft. per minute.

Each furnace is given a 300-lb. charge which is melted at low temperature, to avoid dusting, for three hours. The furnace is rolled from side to side during melting, after which the molten material is poured into 20x13-in. cone-shaped cast-iron pots. When these have cooled, the gold buttons are separated from the matte and slag, weighed, and returned to a furnace to be refined.

Refining Process

This process consists of melting the buttons, blowing air over the surface of the molten metal, and skimming off the lead oxide and other impurities. Time of treatment varies from 5 to 6 hr., after which the gold is poured into cast-iron loaf-shaped molds, having a capacity of 1,500 troy ounces.

Bullion bars are cleaned of adhering slag and matte with a wire brush and dilute hydrochloric acid, and are sampled by drilling at diagonal corners with an electric drill. After sampling, they are weighed on a Fairbanks bullion scale, and boxed for shipment to the Royal Mint at Ottawa, Canada. The bullion has a total fineness of 960, as indicated by the analysis in Table XIX.

The matte is separated from the slag, fluxed with lead oxide, from the gold refining process, and, if necessary, additional litharge is used to bring the total amount of lead oxide up to 15 per cent of the matte weight. This is returned to one furnace, melted, and poured into pots, a gold-silver-lead button being produced. This is refined in the manner described, and shipped as a base bar with a fineness of about 740 in gold and 190 in silver.

Between clean-ups, the matte is again treated in the same way, and a matte bar, low in gold value, is obtained. This is shipped with the next clean-up.

Matte and slag remaining are crushed to about 1 in. by an 8x12-in. Mitchell jaw crusher running at 330 r.p.m. They are then sampled, weighed, and bagged for shipment to a smelter. Usually, a shipment of this material is made once a year.

Furnace linings when removed are crushed in the jaw crusher and then put through a set of 8 in. (diameter) by 5 in. Sturtevant rolls, operating at 290 r.p.m. The rolls, jaw crusher, and blower are driven by a 15-hp., 550-volt, 760-r.p.m. motor, through a common lineshaft. The product from the rolls is approximately 20 mesh. This is run over a 3x4-ft. Wilfley table, driven at 330 strokes per minute, from the same lineshaft as the acid-treating tank mechanism. The

concentrate thus obtained is melted and added to subsequent clean-ups. The table tailings are sampled, weighed, and mixed with the slag for shipment. The yearly production of byproducts amounts to approximately 30 tons and contains less than 0.5 per cent of the total bullion produced.

The refinery crew consists of a chief refiner and a helper. When melting and refining, two extra men are needed.

Cost of refining, based on original tonnage, for labor, employees' insurance, supplies, maintenance, power, and general expense, is $0.0222 per ton (or $0.0496 per ounce of bullion).

The Mill Building

Compact and Fireproof

FOLLOWING THE ORIGINAL PLAN of centralizing all operations, decision was reached to build the mill near No. 11 shaft, inasmuch as the mine ore bin, the cone crusher and auxiliary equipment had already been installed in the west side of the shaft building.

To use a wooden structure for the mill would have necessitated locating it at a safe distance from the shafthouse because of fire risks. This would have entailed higher heating, insurance, and ore transportation costs. Estimates showed that the savings effected in these costs, over a period of a few years, by building a steel and tile structure close to the shafthouse would more than offset the difference in capital expenditure for the two respective types of buildings.

Accordingly, a mill site was selected approximately 150 ft. southwest of No. 11 shaft. and a plant was designed to crush and float a minimum of 2,000 tons daily and to treat 400 tons of flotation concentrate by cyanidation. The flotation and cyanide sections were so arranged that the flow could readily be changed to give any desired combination of treatment, without curtailing output in any way.

Excavations started July 28, 1930, and the work went forward rapidly, the building being housed in by Jan. 1, 1931. Machinery foundations, floors, and equipment were placed during the winter months. Structural steel, amounting to 1,666 tons, was fabricated and erected by the Dominion Bridge Company. Eight-inch hollow tile was used for all walls, as well as for the partition separating the mill from the crushing section, and a built-up roofing laid on transite corrugated asbestos deck was installed by Johns-Manville. The building is 244x176 ft. Details are shown in Fig. 1, p. 473. Cubic content by sections is as follows: Crushing section, 463,220; flotation section, 1,144,819; cyanide section, 865,106; refinery, 88,519; total, 2,561,664 cu.ft.

For repair purposes, the cone crusher, rolls, and tube mills are served by individual traveling cranes. That at the crusher is of Northern Crane manufacture, with a 16-ft. span and a 20-ft. lift. It is operated by a 10-hp. motor and is of 20 tons' capacity. The 22½-ton crane at the rolls is of the Morris double-girder type. It has a 32-ft. span, a 32-ft. lift at 8 ft. per minute, and is operated by a 17½-hp. motor. Above the tube mills the crane is also of the same type, but with a 39-ft. span and 7½-ton capacity. It has a 12½-hp. motor, and a hoisting speed of 15 ft. per min.

Pump floor, flotation, and filter floors are serviced by a 3-ton Otis Fensom elevator, with a 6x8-ft. platform, operating at 66 ft. per min. It is equipped with a double pushbutton control and driven by a 9-hp. motor through a speed reducer.

Table XX—Psychometric Averages, Mill and Crusher House

Jan. 25–Feb. 7, 1932

Location	Dry Bulb Temp. Deg.F.	Dew Point Deg.F.	Relative Humidity, Per Cent
Outside air	6½	5½–6	95 plus
Crusher Building			
Top of surge bin	52	40	64
Top of mill storage bin	52	43	71
Bottom floor	38	32	78
Mill (No steam heating)			
Tube mill floor	57	50	76
Flotation floor	56½	50	78
25 ft. above flotation floor	61½	52	71
Filter floor	61	53	77
Agitator floor	61½	54	75
Pump floor	57	49	75

The plant has sixteen conveyor belts, totaling 2,300 ft., all of 32-oz. duck, varying from 4 to 7 ply and from 24 to 36 in. wide, with ⅛-in. rubber face and 1/32-in. rubber backing. All conveyor idlers and return rollers are equipped with Timken bearings and Zerk lubrication fittings.

Launders in the various distributing systems are of wood, lined wtih Linatex, and vary in slope from 1¼ to 2¼ in. per foot.

Spillage in the flotation section is collected in a central sump, from which it is transferred to the classifiers by a bucket elevator, the belt being 14-in. wide with 46-ft. centers. It is equipped with 66 buckets, each 14x7x7¼ in., and is driven at 360 ft. per min. through a gear and pinion, by a 10-hp. motor with a V-belt drive. A cement wall 24 in. high and 6 in. thick separates the flotation and cyanide sections. This necessitates another elevator in the cyanide unit to return spillage. This elevator is similar to the one in the flotation circuit except for having 50-ft. centers and only 48 buckets.

Duplicate pumps are installed at all points in the flowsheet where a shutdown would be likely to interrupt production.

Flotation cells are equipped with rubber-covered impellers, the rubber being vulcanized to a metal core. This type has been in service for over a year without showing appreciable wear, whereas the original cast-iron impellers formerly used were worn out in less than six months.

All classifiers, triplex pumps, and other machines with bearings at inaccessible points are equipped with Zerk fittings, and are lubricated from central points on the respective individual machines.

Large pieces of wood in the mill feed are removed from No. 1 conveyor by a laborer, while chips and fibrous material are collected on screens located at the overflow of the classifiers.

Standard, all-steel sash ventilating windows are used throughout the building. Ample lighting is secured as well as complete protection from the weather, and no electric lighting is required during daylight hours. Metal ventilating skylights with reinforced glass are installed in the roof.

All floors, foundations for tanks, classifiers, pumps, and other complementary units are of concrete. In fact, aside from the No. 24 Denver flotation cells, the launders, and a flooring over the agitator tanks, no wood is used in the building.

As a safeguard for employees, all stairs, walkways, and machinery are protected with 2-in. steel pipe railings. All V-belt drives are enclosed by screen guards.

The plant is heated, where necessary, by Dwyer unit heaters or standard steam radiators. The heaters are operated by 1/6-hp. 110-volt motors, three of them being used in the crusher house and two in the refinery. The radiators are used in the mill office, laboratory, and that part of the shafthouse occupied by the cone crusher. No heating is required in the flotation and cyanide sections, as sufficient heat is given off by motors and solutions.

Average psychometric data for mill and crusher house for the two weeks from Jan. 25 to Feb. 7, 1932, are given in Table XX. Determinations were made with a wet and dry bulb sling hygrometer, assuming a constant pressure of 30 in.

Power is distributed to the mill as shown in Table XXI.

Table XXI—Power Distribution in Mill

	Horsepower
Crusher plant (except underground crusher)	279
Flotation section (includes five tube mills)	1,516
Cyanide section (includes two tube mills)	558
Refinery	17
Assay office	70
Water supply	33
Heating and lighting	80
Total	2,553

McIntyre's Equipment

Mechanical and Electrical

W. F. J. Cosser

IN KEEPING WITH THE MINE, the mechanical equipment of the McIntyre had a very small beginning. The first hoist and compressor were steam operated, as was also the first ten-stamp mill. When, in 1912, hydroelectric power was made available, 300 hp. was sufficient for the mine's needs. In the early years, lack of funds made necessary many ingenious expedients by the operators of the property. Although many of these practices were of a temporary nature, they served to bring the mine into production.

Growth of the mine, with increased production and development to greater depth, has required, from time to time, the addition of mechanical and electrical equipment. The present centralized plant is equipped with the most modern machinery. Because of the depth attained and tonnage handled, hoisting has been one of the chief mechanical problems.

In selecting and operating the hoisting equipment for No. 11 shaft, the most modern practice was followed. Attention was given to the peak demand method, in local use, of purchasing electrical power. The drive chosen keeps down sudden demands on the power company, and so helps in its operation and lowers the cost of power to the mine.

All ore was to be hoisted in skips from a loading pocket at the bottom of the shaft. Men were to be handled quickly and safely. Two double-drum hoists for balanced winding were accordingly selected. The rock hoist handles in balance two 6-ton skips at a maximum speed of 3,000 ft. per min. The cage hoist handles a 60-man double-deck cage in balance with a counterweight, at a maximum speed of 2,000 ft. per min.

No. 11 shaft is over three-quarters of a mile deep. Hoists and cables must provide maximum safety at the speed and with the loads in use. The need for choosing safe, powerful hoists and hoisting equipment and cables capable of standing the most exacting service is apparent. The importance of day-after-day inspection and maintenance of all of these can also readily be understood. General specifications of the

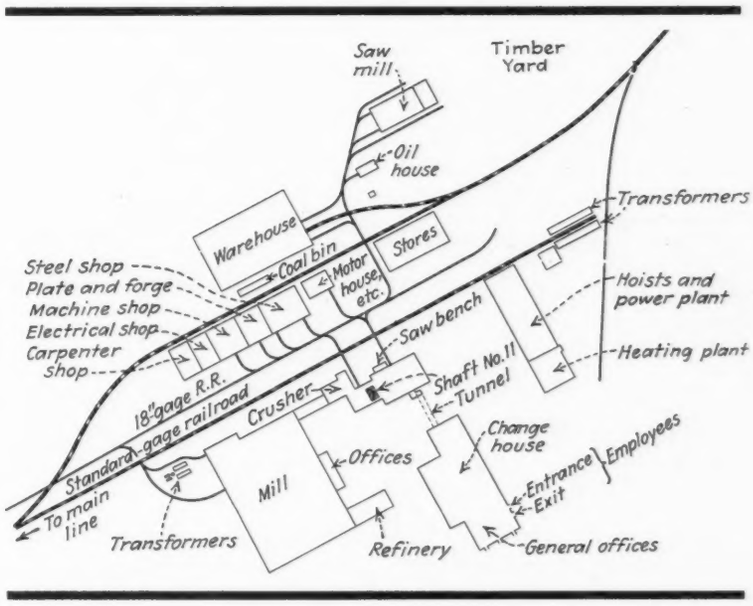
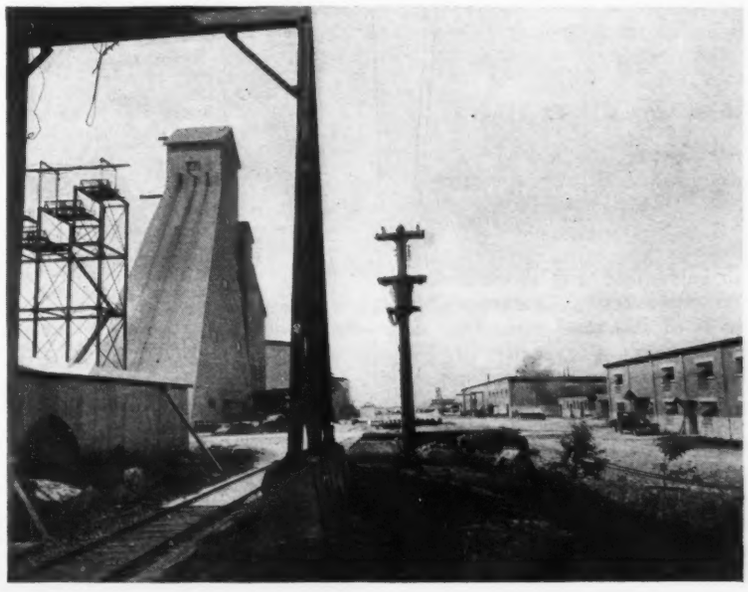


Fig. 1—Layout of centralized surface plant at No. 11 shaft

Shops of centralized plant at No. 11 shaft



hoists and ropes which handle this work are given in Table I.

Hoist specifications were drawn up by the mine's mechanical staff, the members of which also erected the units. Operation and maintenance of these hoists, the ropes, the sheave wheels, the cage and skips in the best running order is one of the most important duties of this department.

Ropes are looked over daily and examined thoroughly once a week. They are measured and greased once a month. All ropes have a piece cut from the end for Government test every six months.

The Hoists

The rope drums are 12 ft. in diameter and 8 ft. face with a 300 ft. rope lead over a sheave tower to the shaft sheaves. The 1¾-in. diameter cage and counterweight ropes have 2⅓ layers on the drum. Their life is 36 months. Some crushing develops on the idle turns on the drum; 80 to 90 ft. of rope is therefore cut from the drum end at half-life of the rope. When the rope is finally taken off, the Government breaking tests show little or no falling off in strength. The core is becoming dry and there is evidence of fatigue in individual threads and some reduction in diameter. The 1½-in. skip ropes have two layers on the drum. Ropes now in use have hoisted 689,307 rock-tons each. Previous ropes, the first used, hoisted 513,300 rock-tons each over a longer period and were taken off owing to one rope basketting at about 200 ft. from the skip. Every day each skip travels a distance of 300 miles.

Shaft No. 11 hoists are housed in a building 166 ft. long, 56 ft. wide, and 42 ft. high, with a basement for the lower part of the hoists and for electrical controls. The building is steel frame with walls of hollow tile, plastered outside. Its roof is gypsum slab with laid-up 4-ply covering. The floor is 1-in. thick mastic on concrete. A 45-ton, overhead crane with one motor and hand transverse spans the building and services its whole length. Heating is by Aerofin units.

Headframe 175 Ft. High

The headframe is of heavy steel construction and has an over-all height of 175 ft., with the sheaves 140 ft. above the collar. The frame is sheathed with 1½-in. T & G plank, covered on the outside with galvanized iron sheets and on the inside with asbestos sheathing. The main bin is of 700 tons' capacity. An auxiliary bin has a capacity of 350 tons. The sheaves are 14 ft. in diameter and the idler tower is 80 ft. high.

Cage and Skips—The double-deck cage weighs 7½ tons and carries 60 men, or steel or timber trucks 11 ft. long.

Two spare 6-ton skips insure continuous operation. Some difficulty has been had with fine ore packing in their bottoms. After trial of various schemes, a line of heavy chains on the bottom and fastened at one end only has relieved the situation.

A 12x16-in. air hoist on the 1,250-ft. level operates a service cage from the 1,325-ft. level to the bottom. It carries four men and is much used for carrying small supplies.

Data for the hoists at Nos. 5, 6, 7, and 12 shafts are given in Table II. The permanent hoist at No. 12 subshaft was manufactured by Fraser & Chalmers Company, makers of the larger surface hoists at No. 11 shaft. It will handle, in balance, combination skips and cages.

Table I—Hoisting Equipment, No. 11 Shaft

Hoists—Two Fraser & Chalmers, Witton, England, main, first-motion hoists exactly similar in mechanical details, cast-steel drums (6 sections) 12 ft. diameter by 8 ft. cylindrical (one fast, one loose). Rope grooves parallel.

Drum Shaft with half coupling integral, M.O.H. steel, 21 in. diameter with 4-in. hole bored full length of shaft.

Clutches—Cast-steel multi-jaw external tooth, sliding on hexagon, operated by air engine.

Brakes—Suspended, parallel motion, gravity with automatic take-up, operated by air engine with oil cataract cylinder.

Brake Paths—13 ft. 6 in. by 1 ft. 2 in., specially ventilated.

Brake Lining—Ferodo, ¾ in. thick.

Bearings (3)—Swivel type, oil-ring lubrication; center bearing with wedge adjustment.

Motor Bearing (1)—Swivel type, oil-ring lubrication.

Control—Ward Leonard Ilgner for rock hoist. Ward Leonard for cage hoist. 52-ton flywheel on rock hoist converter set. Provision for addition of flywheel to cage-hoist converter set if it should become necessary.

Safeties—Whitmore overwind and overspeed accelerating and decelerating cams governing driver's lever; hatchway switches and all standard electrical accessories; brakes operate automatically in case of failure of current (d.c.), overload, overspeed, or overwind. Application of brakes is gradual in working portion of shaft, and rapid when approaching tips or in case of overwind.

Interlock—Provided to prevent release of brake from an unclutched drum.

Indicators—Eight feet in diameter, with approach alarm bell, and supplementary pointer to give amplified indication over final 150 ft. of wind.

Auxiliary Compressors—Two direct-coupled, 50 cu.ft. capacity each, 100 lb. per sq.in., for use in case of low pressure from mine supply.

Duty—Balanced winding. Depth of shaft 4,125 ft. vertical. Rock hoist 6-ton capacity, self-dumping skips. Cage hoist, double deck, cage 60 men, and counterweight.

Electrical Equipment for Hoist

(General Electric Company, Witton, England)

Rock Hoist—Motor: 2,450 hp., 79 r.p.m., 550 volts, d.c.
Flywheel set: generator 2,000 kw., 500 r.p.m., 550 volts.
Induction a.c. motor, 1,700 hp., 500 r.p.m., 2,200 volts, 23 cycle with slip regulator.
Exciter: 35 kw., 500 r.p.m., 220 volts, d.c.
Flywheel: 52 tons, two main bearings, oil pressure 175/190 lb. per sq.in. provided by four gear wheel pumps, two direct-coupled to 5-hp., a.c. motor for use when starting up, and two belt-driven from flywheel shaft for use when running.

Cage Hoist—Motor: 1,400 hp., 52.5 r.p.m., 550 volts, d.c.
Generator: 1,200 kw., 500 r.p.m., 550 volts, d.c.
A.C. Motor: 1,700 hp., 500 r.p.m., 2,200 volts, 25 cycle.
Exciter: 35 kw., 500 r.p.m., 220 volts, d.c.

Ropes—Speeds, 2,000 ft. per min. for cage hoist (A), 3,000 ft. per min. for rock hoist (B).
Diameter and construction: (A) 1¾ in., 6/28 flattened strand, Lang lay. (B) 1¼ in. 6/17 Lang lay, round.
Rope on drums: (A) 2⅓ layers, (B) 2 layers.
Factors of safety: (A) 6 plus. (B) 5.5 plus.

Table II—Hoisting Equipment, Shafts Nos. 5, 6, 7, and 12

No. 6 Shaft (3,015 ft. deep).
Hoist—Nordberg 7x4-ft. cylindrical, double-drum, single-reduction geared, single-clutch hoist.
Motor: 350 hp., 550 volt, 25 cycle, Canadian Westinghouse Company.
Speed: 1,500 ft. per min., 1¼-in. diameter ropes, 7-ft. sheaves.
Present Duty: Men and material, cage and counterweight.
Past Duty: 3-ton ore skips.
Safety: Lilly.

No. 5 Shaft (2,385 ft. deep).
Hoist—Nordberg compound-drum, 6 ft. and 5 ft. in diameters, single-reduction geared hoist.
Motor: 175 hp., 550 volt, 25 cycles, Canadian Westinghouse Company.
Speed: 1,150 ft. per min., 1-in. and ⅞-in. ropes.
Duty: Men and material, cage and counterweight.
Safety: Welch.

No. 7 Shaft (1,000 ft. deep).
Hoist—Webster Camp & Lane, single-reduction, double-clutched, double-reduction geared hoist.
Motor: 200 hp., 550 volt, 25 cycle, Allis-Chalmers Bullock Company.
Speed: 650 ft. per min., 1-in. ropes, 5-ft. sheaves.
Duty: Men and material, cage and counterweight.
Safety: Lilly.

No. 12 Sub-Shaft (to have final depth of 3,000 ft. below 3,750-ft. level).
Hoist—Preliminary sinking hoist used for first 700 ft. was an 11x8-in. stroke, Canadian Ingersoll-Rand, SSR, single-drum air hoist, ¾-in. flattened strand, non-rotating rope.
Hoist—Permanent winder. Double-drum (cylindrical) 8 ft. diameter x 4 ft. face.
Clutches: Two, multi-jaw external tooth, sliding on quadrant. Engines, oil-operated.
Brakes: Suspended parallel motion. Engines, oil-operated.
Gearing: Single-reduction helical.
Ropes: Preliminary, ¾-in. flattened strand, non-rotating. Final, 1¼ in.
Speed: Preliminary, 1,500 ft. per min. Final, 2,000 ft. per min.
Final depth: 3,000 ft. vertical.
Controller: Liquid, Allen West Company of Canada.
Motor: 800 hp., 2,200 volt, 25 cycle, English Electric Company of Canada.
Safeties: Lilly.

Surface Transportation—A standard-gage spur line connects the plant with the T. & N. O. Railway, Porcupine Branch. Eighteen-inch gage track, 30-lb. rail, minimum radius 30 ft., serves shafts, shops, warehouses, mill, and yards. Two 3-ton Whitcomb gas locomotives handle this work.

Compressor Plant—The mine compressor plant, total capacity of 10,200 cu.ft. of free air per minute, is located centrally to the underground work close to No. 6 shaft. The equipment is given in Table III. This unit system has permitted addition of new compressors as required.

All compressors have plate valves. The Ingersoll-Rand compressors have 3- and 5-stage unloading. The Bellis & Morcom and Alley & McClellan machines are controlled by cut-off intake valve. All intakes are screened, inlet boxes of ample area, covered with wire netting and two layers of cocoanut matting, being used. Cooling water is pumped from Pearl Lake.

Air mains are 10 in. to No. 11 shaft; 8 in. to No. 6 shaft, and 6 in. to No. 5 shaft. Air pressure is 90 lb. per sq.in.

Shops Adequate

Shops—As part of the mine's centralization plan, a shop containing carpenter, electrical, machine, plate, and steel shop sections was built in 1931. Its construction is steel-frame, hollow-tile walls with laid roof on Haydite slabs. All shops, except the carpenter shop at the end, are interconnected by 18-in. tracks, with fire doors in the party walls. Carpenter, electrical, and machine shops are floored with wood blocks. Plate and steel shops have dirt floors. Machines are served by overhead crawl beams. Heating is done by Aerofin units and low-pressure steam from the central plant.

The shop handles all repairs and makes certain replacement parts, including classifier rakes, tube-mill scoops, and the like, for the mill, and truck bodies, drag scrapers, diamond- and rock-drill repair parts, locomotive general repairs, and similar work for the mine. Much of the fabricated steel used in recent construction work has been made up in the plate shop.

The equipment includes new units, as well as units in good order moved from the old shops. New units have individual motor drives. Old equipment is belt-driven from countershafts.

In the steel shop, an average of 2,100 steel is being sharpened in two 8-hr. shifts, in addition to making up scaling bars, gads, moils, picks, and chute irons. The section used is 1-in. hollow octagon (Atlas) made up into cross bits (14 deg. and 3½ deg.). Seven steps are used from 1¾-in. starters 2 ft. long to 1⅜-in. steels 11 ft. long, the steps or changes being 1 ft. 6 in. in length and $\frac{1}{16}$ in. in bore. Steel is all block-gaged, the dollie diameter being ⅛ in. over size of finished bit.

The sharpening furnace temperature is 940 deg. C. and that of the tempering furnace 875 deg. C. Both furnaces are coal-fired. In the O'Donovan-type tempering furnace, the steel is advanced by hand from the rear end of the furnace through a series of cast-iron muffles and Carbofrax tubes holding individual steel. In front of the last two tubes the steel rests on counterweighted withdrawing gear. When the steel is raised to the tempering point or point of decalescence at 775 deg. C. it loses its magnetism. The field of a magnet set at the back of the last two tubes and holding the steel in the furnace is then overcome and the steel is withdrawn from the furnace at a uniform tempering heat by the counterweighted gear.

These O'Donovan sharpening and tempering furnaces were installed April, 1925, and from the start gave a large reduction, at that date about one-fourth, in the number of steel required for footage drilled. Shop costs, including supplies, average 5½c. per point handled.

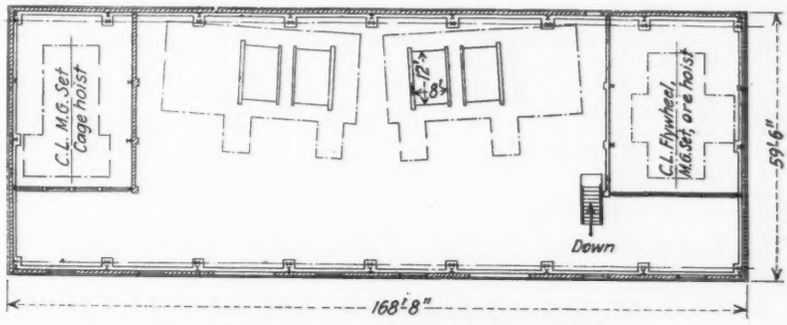
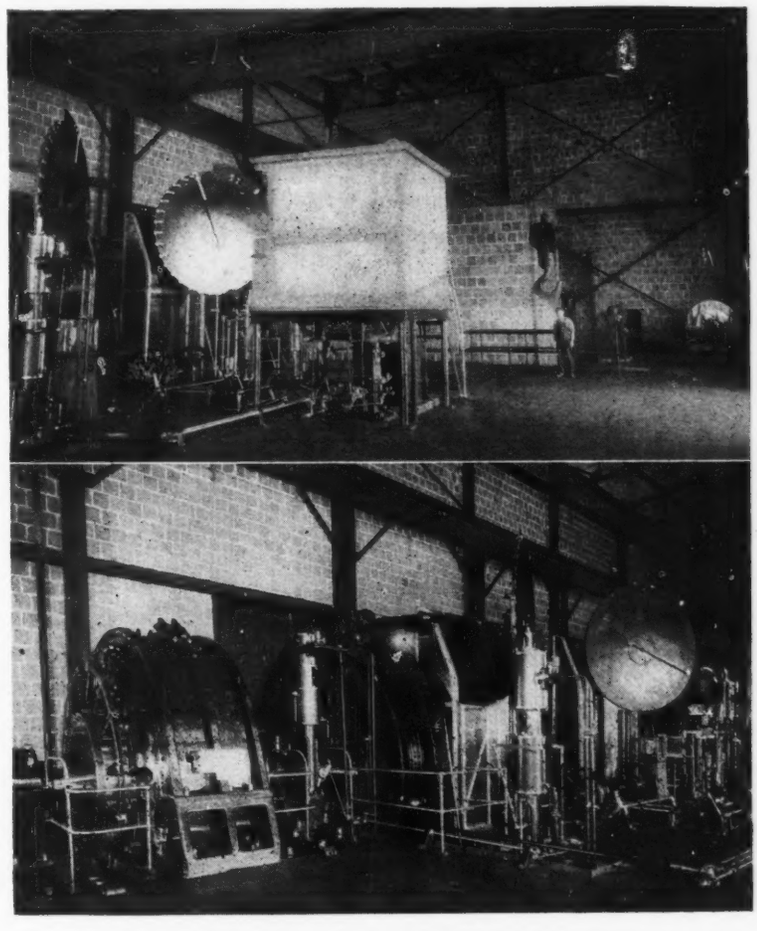


Fig. 2—Floor plan of hoist house at No. 11 shaft



Hoist room at No. 11 shaft, showing ore hoist (above) and man hoist (below)

Water Supply—Pearl Lake, on the property and within 500 ft. of the plant, is the main source of water supply. Pump equipment is listed in Table V.

Mine Pumping—The amount of water dealt with is small, being only 220,000

Table III—Equipment of Mine Compressor Plant

1 Ingersoll-Rand PRE 2, 2,800 cu.ft. per min. (after-cooler) direct-driven, 480-hp. synchronous motor, 175 r.p.m.

1 Bellis & Morcom, 3,000 cu.ft. per min. direct driven, 600-hp. synchronous motor, 250 r.p.m.

1 Ingersoll-Rand PRE 2, 3,200 cu.ft. per min. (after-cooler) direct-driven, 600-hp. synchronous motor, 250 r.p.m.

2 Alley & McClellan 600 cu.ft. per min. each direct driven, 125-hp. induction motor, 365 r.p.m.

3 Receivers at plant (1) 4x12 ft.; (2) 5ft. x 14 ft. 6 in.
Receivers at all shaft heads, and where required underground.

Table IV—Shop Equipment

Machine Shop—60x60 ft.	3 lathes, 3 drill presses, 1 bolt threader, 2 pipe threaders, hack saw, 1 heavy-duty shaper, 1 hydraulic press, emery wheels.
Plate Shop—45x60 ft.	One punch and shearing machine, one motor-driven pneumatic air hammer, one bending rolls, one electric welding machine, oxy-acetylene torches, and related appliances.
Electric Shop—37 ft.6 in. x 60 ft.	Bench lathe, two drill presses, and emery wheels.
Carpenter Shop—37 ft. 6 in. x 60 ft.	Planer and moulder, band saw, cut-off swing saw, rip saw, jointer planer, and blade-sharpening machine and saw gummer.
Steel Shop—80x60 ft.	Three No. 550 Leyner sharpeners, on which all steel is sharpened and new steel made up and shanked. One Leyner sharpener, with special blocks and an extended foot treadle, for straightening bent steel. One Leyner sharpener for making up arm bolts, chute bolts. One Leyner sharpener carried in reserve. Converter set for supplying d.c. current to magnet. Blackmer belt-driven power hammer. Emery wheels. Special trucks for transporting and lowering steel on the mine cage of No. 11 shaft.

Table V—Pump Equipment, Surface and Underground

Mill
Two DeLaval pumps, one in reserve, 700 Imp.g.p.m. direct-coupled to 40-hp., 550-volt, 25-cycle, 1,450-r.p.m. induction motors. Pressure per sq.in., 65.

Hoist Water-cooling System
Two Goldie & McCulloch, one in reserve, 400 Imp.g.p.m., direct-coupled to 30-hp., 550-volt, 25-cycle, 1,450-r.p.m. induction motors. Pressure 70 lb. per sq.in.

Mine
1,000-ft. level—One 7x12-in. Evans horizontal, triplex, belt-driven by 2,200-volt, 500-r.p.m., induction motor.
1,875-ft. Level, 2,875-ft. Level, 3,875-ft. Level } Each, 5x10-in. Aldrich vertical triplex, belt-driven by 2,200-volt, 500-r.p.m. induction motor.

Chain feeder of underground jaw crusher below 3,875-ft. level, No. 11 shaft

gal. per 24 hr. On this account plunger pumps have been adhered to. Adequate sumps are provided. At No. 11 shaft, where all of the water is pumped to surface, pump stations are located at the 1,000-, 1,875-, 2,875-, and 3,875-ft. levels, with a small unit at the shaft bottom (see Table V). Each delivers to the pump station above it, with the 1,000-ft. level pump discharging to surface through an 8-in. line.

Electricity's Part

Electricity has been responsible for much of the advance made in Ontario mines. The McIntyre has used hydro-electric power since 1912. The present load is 6,000 hp. with 440 motors in use. Power at 25-cycle frequency is purchased, as by the other mines of this part of northern Ontario, on contract from Northern Canada Power Company. In 1921, an increase in mill capacity was hindered by a power shortage during late winter months. The company acquired the rights to develop Sturgeon Falls, on the Mattagami River. These rights were assigned to the power company on terms that amply safeguarded the McIntyre power requirements. The supply of hydro-power from the company named in the foregoing, as shown by the accompanying power-supply chart, is not dependent on one generating station alone. Interruptions of supply are rare.

The Schumacher sub-station, where the power company's systems are tied together, is located on the McIntyre property between the main No. 11 shaft on the east and the No. 6 shaft and compressor plant on the west. Power is taken by the mine from it at 12,000 volts. All power is metered by the power company on a monthly peak load basis.

Details of electrical equipment presented in the following have been furnished by F. W. Gilbert, electrical engineer.

MINE POWER is distributed to three transformer stations, where it is stepped down for general use at 550 volts. At No. 11 shaft sub-station, however, four transformers of 1,000 kva. each have 2,200 volts secondary for No. 11 converter set motors, mill crushing plant, and underground at No. 11 shaft.

No. 11 sub-station is the source of supply for the two complete hoisting equipments at No. 11 shaft. All of the electrical apparatus, including driving motors, motor-generator sets, switch and control apparatus, is made by General Electric Company, England.

The reversing, shunt-wound, compensated d.c. motors for the rock and cage hoist have normal outputs of 2,450 hp. and 1,400 hp., respectively. Their frames are ribbed to prevent deflection, which gives additional stiffness to the machines as a whole, both being built in the same size of frames. Armatures

are banded by means of steel straps and cotter pins to eliminate troubles from banding wires.

The Ilgner motor-generator set for the rock hoist is assembled in the following order: 1,700-hp., a.c., 2,200-volt, 500-r.p.m. at synchronous speed, 25-cycle motor; 50-ton flywheel; 550-volt, 2,000-kw. generator; and 220-volt, 35-kw. shunt-wound exciter. This generator is of the compensating and high-reluctance commutating pole type, mounted on a common shaft and bedplate with motor and flywheel. Bearings are of spherical-sleeve, water-cooled, oil-ring lubrication type, which facilitates alignment. Those on the flywheel have water-cooled oil admitted to bottom at 200-lb. pressure, which floats the shaft on an oil film. The generator commutator has a 1 ft. 7 in. face and 3 ft. 4 in. diameter with a peripheral speed of 5,239 ft. per min.

The flywheel has a peripheral speed of 21,000 ft. per min. and will run for 1¾ hr. after power has been switched off. On failure of power, it can complete two full winds with the loaded skip. It has a 60 per cent equalization and a 10 to 15 per cent drop in speed on full-load hoisting cycle. For shutting the set down, an emergency brake is provided which takes 20 min. to bring the wheel to rest. The set can be brought to full speed in 5 min.

For control of the a.c. motor of the set, slip-regulator electrodes are mounted in porcelain pots inside the tank. Movable elements are operated from above from the torque motor installed in the base. Owing to the extended movement of these elements, the starting peak can be well controlled. The electrolite is kept at an operating temperature by a water-cooling system.

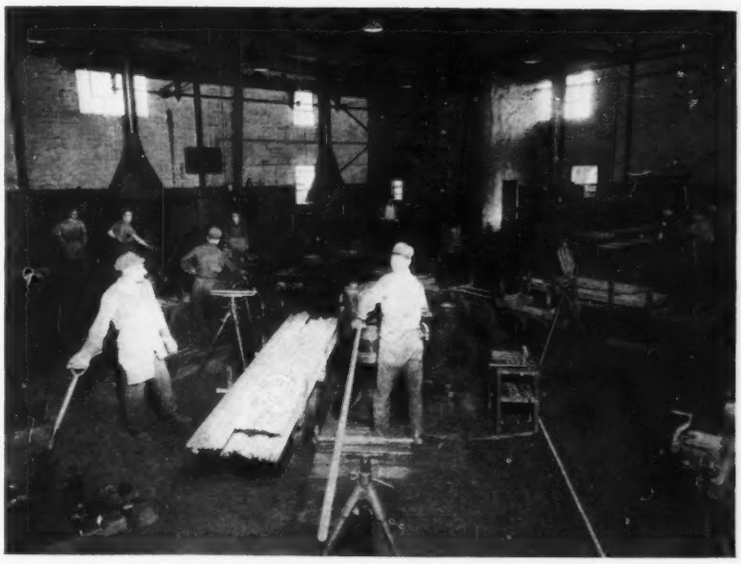
The Cage-hoist M-G Set

The equipment of the cage hoist, Ward Leonard motor-generator set, is a duplicate of that of the rock hoist, except that it has no flywheel, torque motor, or underload relay on the high-tension equipment. Provision for these, however, in case of need, has been made in the equipment and foundations.

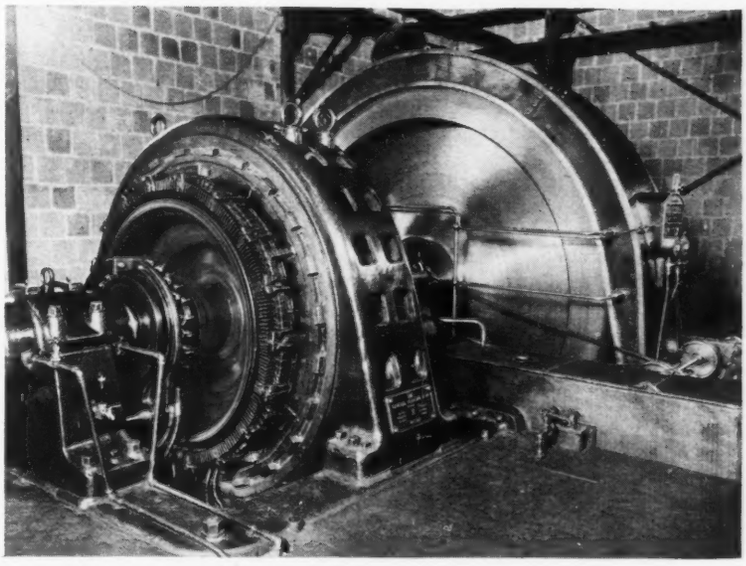
The a.c. motor is a duplicate of that of the other set, so as to minimize the number of spare parts. Generator output is 1,200 kw. normal capacity, and its exciter is 35 kw.

For control of the generator field, a series regulator has been used. The regulator contacts themselves have small currents to make and break, which assists in operating efficiency. Also, two contacts in parallel are used for introducing a weakening resistance in the hoist motor field at the end of each wind and to avoid use of contactors for this purpose.

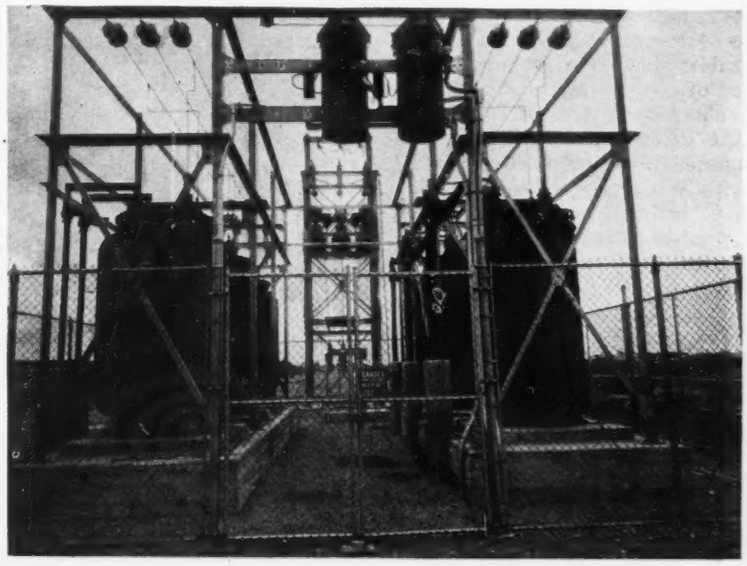
This regulator is also provided with contacts for operating a suicide contactor at the end of each wind after reducing the field strength to a very low



In the steel-sharpening shop at No. 11 shaft



Motor-generator set, with flywheel, for No. 11 shaft ore hoist
Below—Outdoor transformer station for the mill



value. The controller consists of a face-plate type of reversing shunt-field regulator with resistance and having 50 operating contacts for each direction of rotation, coupled with sector and pinion, with shaft for coupling to the drivers' operating lever. All contactors, exciter rheostat, and relays are mounted on one panel.

The motor-generator rooms (partitioned off at the end of the hoist room) are each ventilated by centrifugal blowers—5 hp. for the cage-hoist set and 10 hp. for the ore-hoist set, the piping being extended to and above the generators.

Hoist men's instruments of the illuminated type are used, comprising speed indicator for the motor-generator set and ammeter and voltmeter for main d.c. circuit. A suitable alarm device is fitted to warn the driver against overspeed of motor or generator set.

The signaling system for No. 11 shaft hoists consists of an electrically operated hooter and pull to signal the hoist man and a hooter and pull for the call of the cage on every mine level. From the collar of No. 11 shaft to the hoist man there is also an individual signaling system. Northern Electric, wall-mounted, wood-case telephones with pushbuttons and extension bells for 110-volt ringing are mounted in each shaft station. No. 11 signaling cable, known as McIntyre special (Northern Electric Company), is used.

There are three main 2,200-volt power cables down No. 11 shaft, two composed of No. 1 stranded, three-conductor rubber-covered with $\frac{3}{64}$-in. lead and No. 6 galvanized steel armor, and one composed of 4/0 stranded and insulated with $\frac{9}{64}$-in. varnished cambric, $\frac{3}{64}$-in. V.C. belt, $\frac{3}{64}$-in. lead and No. 6 B.W.G., galvanized-steel wire armor. The latter cable was recently put down the shaft to the 3,750-ft. level and extended 1,800 ft. along the level to the new internal shaft (No. 12). The three power cables are served every 25 ft. for 2,200-volt underground system built to 3,300-volt standard. All cables are supplied by the Standard Underground Cable Company, Hamilton, Ont.

Power levels for the cables are established at the first, or 1,000-ft., level and every subsequent 500 ft. in the shaft. At these levels, stations for transformer cubicles or link houses are cut out of the solid rock. Similar cubicles are in use at each shaft station for signaling and other cables.

Electric lighting is installed only at the individual shaft stations and is served by 2,200-volt, 110-volt transformers on the power levels.

At various stations, pumps and motors for motor generators to serve charging sets of battery locomotives are operated by 2,200-volt motors.

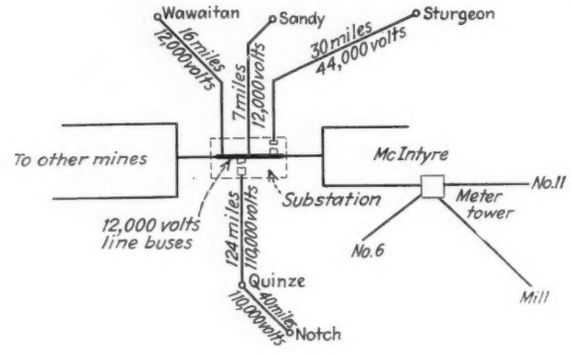


Fig. 3—Power supply chart for Northern Canada Power Company's Schumacher sub-station for McIntyre Porcupine Mines

On each of the 1,875- and 2,800-ft. levels there is a 4-ton Goodman trolley locomotive. The two units, 3-bearing M.G. set for these consists of 40-kw., 750-r.p.m., 250-volt, compound-wound generator, direct connected to and on the same base with 60-ft., 2,200-volt, 3-phase induction motor. The control trips out the d.c. breaker on failure of the a.c. supply voltage.

In No. 11 shaft there are approximately 15 miles of cables for power, signals, auxiliary signals, telephones, and light. Safety precautions against fire have had careful consideration in the installation, in shaft and mine, of all cables, link-houses, motors, and electrical equipment.

The Fraser & Chalmers hoist on the 3,750-ft. level, operating the new internal shaft (No. 12), is driven by an English Electric Company, 800-hp., a.c. motor with an Allen West liquid control and Canadian General Electric Company reversing contactors. The counterweighted, movable electrodes of the liquid control are moved vertically by a compressed-air engine into individual jars made of a special insulating compound, which hold the three stationary electrodes. A special bronze piping cooling-water system is used.

Owing to the position of this internal (No. 12) shaft, safety precautions against fire hazard have been given first consideration at all points of the electrical installation, especially in the case of all electric cables. The heating effect of motor and control has also been taken into account. This is minimized by the construction, and they are placed in the return air current from No. 12 shaft.

A new type of fireproof, signaling cables is being placed in this new shaft. They are five-conductor cables composed of four conductors No. 8 and one conductor No. 14, rubber-covered and with rubber filler, over all a lead cast rubber belt with lead of $\frac{3}{64}$-in. left on, and for 600-volt operation. The system will be split into two parts: signaling and one side of phone, and call system and other side of phone. The advantage will be in size of conductors and fusing of mains. The cross induction from signaling to telephone system will be at its minimum with improved talking conditions and facilities for locating trouble.

Special motor-starting equipment is used for the 2,200-volt motors of underground crusher, the surface crusher and rolls, and for the 550-volt motors of the tube mills and the mill air compressors, which have similar panels except for the control transformers, which differ for the 550 and 2,200 volts.

As the equipment is in a dusty atmosphere, it is enclosed in steel, dead front cases with double doors in front and back. Starters, with their contactors and relays, together with their necessary busbars and connections, are mounted on a compound base supported on angle-iron framework. The control transformer is mounted in the back of the panel on supports. The whole forms a complete, six-point starting unit which provides automatic starting in six balanced steps, whereas the normal

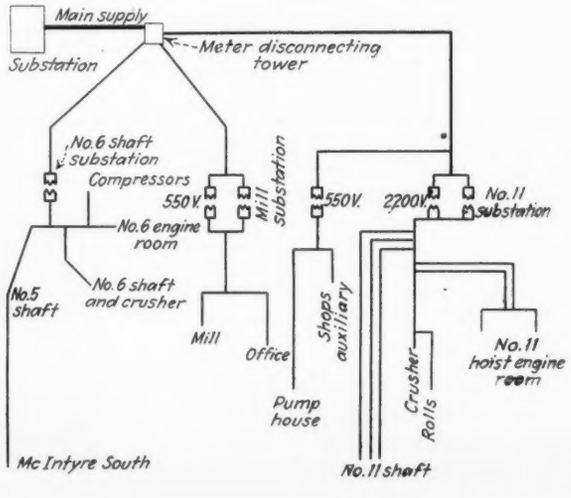


Fig. 4—Chart showing power supply at McIntyre Porcupine Mines

standard starters have four points only.

In this installation the first three steps are governed by relays, each of which can be set for a maximum of 15 sec. The last two steps are governed by time interlocks which can be set for 8 sec. maximum. After the pushbutton is pressed, the line contactor closes and each accelerating contactor closes in succession, cutting out the resistors in given steps. The maximum accelerating period is one minute.

When the final accelerating contactor closes, it lights a red lamp and an interlock on this contactor drops out the intermediate accelerating contactors, consequently cutting down the power requirements of the panel to a minimum.

Overload protection is provided by temperature overloads.

The switching house for the mill sub-station is in a separate room in the mill building, giving a compact installation and a ready distribution of power throughout the mill. This is done mostly in conduits buried in the concrete floors. With the exception of the crushing plant, which takes 2,200-volt current, this sub-station supplies all of the mill power and that for the near-by refinery and administration building, with its mine offices, assay office, and change rooms.

Starters for the motors using this power are standard equipment, except for the special six-point starters described in the foregoing, and other special starting and control equipment.

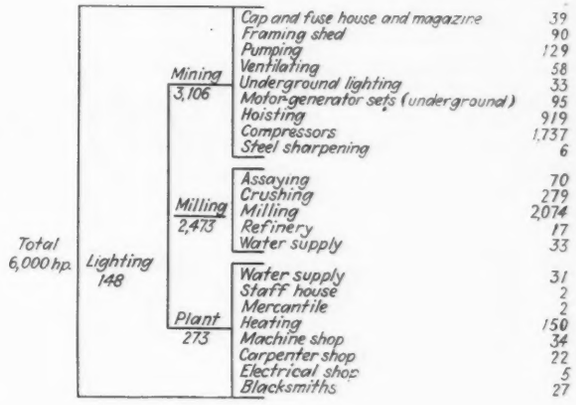


Fig. 5 — Distribution of power consumed by McIntyre Porcupine Mines in 1933. Based on paid horsepower.

No. 6 Sub-Station—At No. 6 Pearl Lake plant, the mine's five compressors and the hoist are operated. No. 6 shaft lighting and signals, with their link houses, are identical with those for No. 11, excepting that 550-volt power is taken off at every level. The power is supplied over a No. 0 cable, three-conductor rubber-insulated, lead-and-steel wire-armored, 600-volt working pressure.

The signal cable is seven-conductor composed of five conductors, seven-strand No. 6, and two-conductors seven-strand No. 14 rubber-insulated lead-and-steel wire-armored. The same type of cable is used in No. 5 shaft for signaling. There is no power in this shaft.

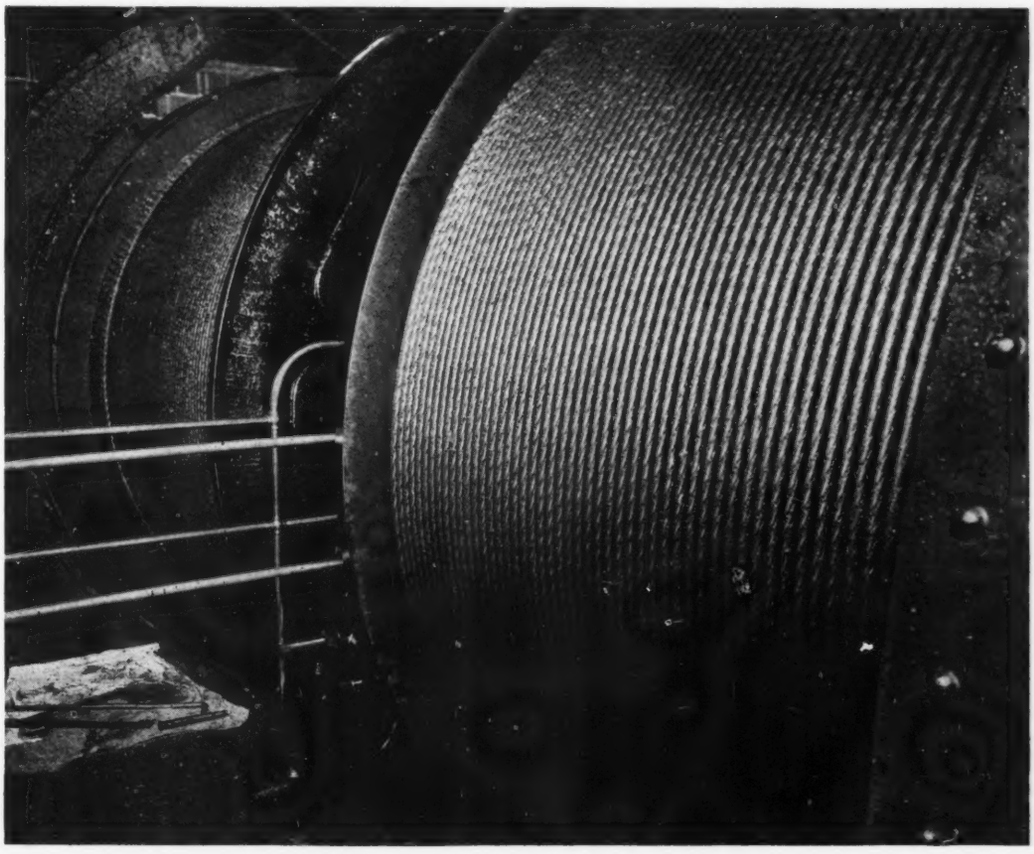
This sub-station is also connected to a standby crusher plant at No. 6 and No. 5 hoist, which has a 175-hp., 725-r.p.m. Westinghouse hoist motor with liquid rheostat.

Power is supplied to No. 5 shaft over two three-conductor, rubber-covered 4/0 lead-and-steel-tape armored cable placed underground.

The total peak of power used at the mine monthly is about 6,000 hp., distributed as shown in Fig. 5.

The mine's electric power supply is ample. The sub-stations are sufficient for further expansion. Equipment is of the type which has proven, over years of heavy mine service, suitable to conditions. Interruptions to production due to electrical failures are few.

Drums of the cage hoist, No. 11 shaft



Safety and Health

Major Considerations at the McIntyre

A. D. Campbell

ADEQUATE ORE RESERVES, with an up-to-date plant and the necessary equipment, are essential to a mining company's success, but they, in themselves, are not sufficient. Success still depends upon the ability and enthusiasm of the employees.

Maximum ability cannot be displayed under dangerous conditions, and enthusiasm is largely the product of a contented mind. Accident prevention, and the miner's health and home life, have always been a matter of real concern to the McIntyre management. The importance this bears to the actual cost of gold production can be readily seen, inasmuch as the cost of labor represents over 60 per cent of the total mine expenditures.

McIntyre takes pride in its interest in its employees and has retained the confidence, good will, and loyalty of the men to a remarkable degree. There is no company town. The community has developed as an independent, self-governing Canadian municipality. Community life and municipal government have had, without any paternalistic attitude, the ready support of the company and of its officials.

In Ontario a municipality cannot tax the mining plant or the ore reserves of a mine, but it receives 1½ per cent of the profits assessed under the Provincial Mines Profits Tax Act up to a point where this nets the municipality $35,000 in a year. On profits over this, the municipality receives 1 per cent. With these substantial receipts and local taxation, the community has financed waterworks, lighting, paved streets, and excellent public, high, and technical schools.

The mine and adjacent towns of Schumacher and Timmins have had railway accommodation since 1912. The community has thus always been within easy reach of the eastern Canadian and United States cities. Employees are predominantly Canadian and British born.

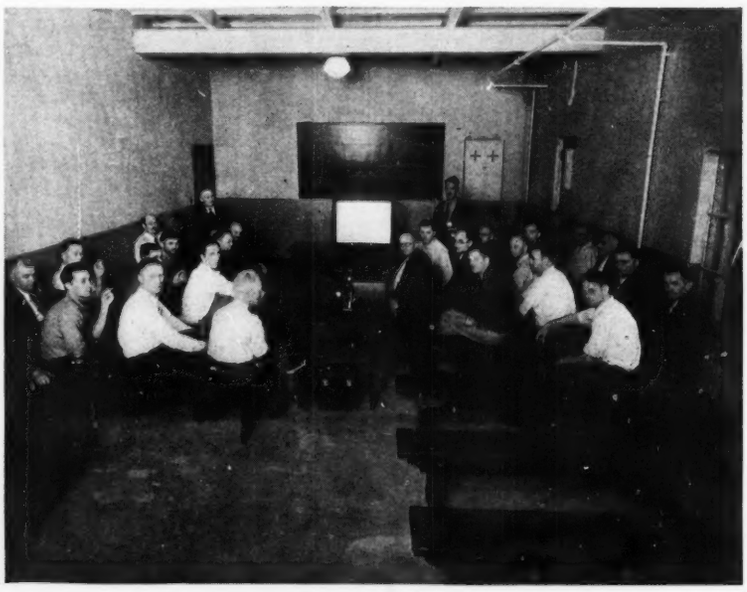
The wage schedules are higher than those of other industries of the district. Underground wages are practically all at two standard rates, $4.24 and $4.80 per 8-hr. day. Most of the underground work is done on contract with the standard wages guaranteed. All men on the contract share in it. During the past year, 236 men on underground contracts made an average of $1.50 per shift in addition to their guaranteed wages. The system works satisfactorily for mine and men and has been of great help in getting the men interested in the more efficient organization of their particular underground work.

At the McIntyre property the interests of labor as exemplified in fair treatment, wage schedules, insurance, health, safety, and accident compensation have been admittedly well looked after. Wage schedules were not reduced during the recent depression. Instead, heavy expenditures were made to benefit health, safety, and general living conditions. These are done without expectation of thanks, but there has been built up a McIntyre spirit and healthy pride of accomplishment which is shared by every McIntyre man and which is a real factor in the story of technical and financial progress recorded in the pages of this issue.

Athletics—Athletics are encouraged as recreation and health-building exercise. Each employee makes a voluntary contribution of 50c. per month to the McIntyre Athletic Association. The executive of this association, with the company physician as active chairman, controls the finances and correlates the activities of the employee clubs which manage the various sports. During the past year the association, with some aid from the company, has built beautifully grassed, athletic grounds with bleachers and grandstand. This was constructed on the tailings which filled the west end of Pearl Lake. The field has separate playing spaces for football and baseball. Mine teams in these sports compete in regular leagues and have been successful this year in winning district honors. Other athletics are confined mostly to interdepartmental competition.

Insurance—Group life insurance is carried by the company for all employees. The policies issued on the completion of three months employment for $500 each are increased by $100 per year to a maximum of $1,500. Complete permanent disability and death are covered.

Underground supervisors at a safety meeting in the conference room, in the change house and office buildings, No. 11 shaft



Medical and Hospital Attention—Medical and surgical needs of the employees are looked after by Dr. W. D. Robson and his assistant, employed by the mine. Towards medical and hospital care for sickness and accidents not covered by compensation, the employees each pay 90c. per month. Of this, 50c. per month goes to the St. Mary's Hospital, of Timmins, as a contract payment. This modern hospital takes care of the Porcupine camp. The mine doctors, in addition to town and hospital offices, have an office in the McIntyre Administration Building, which houses the miners' change-rooms. Here they and the night first-aid man see the shifts come and go to work.

All applicants for work are examined for silicosis by the doctor of the Ontario Compensation Board, and also by the mine doctor for pulmonary diseases and general fitness. Yearly examinations of underground men are also made by the Government doctor, chest X-ray pictures being taken at every examination. Silicosis here is only made serious by tuberculosis complications, so that with T.B. suspects and other unfit applicants not allowed to start underground work, the incidence of silicosis and its complications are lessened.

The mine doctors cooperate definitely with the mine supervisors, this resulting in the early treatment of injuries and early return of injured men to suitable work, with usually a complete rehabilitation of the men to their old jobs. Thus compensation costs to the mine and time losses to the men are lessened.

A Champion Team

First Aid—Medical treatment is supplemented by training in first aid, given to numbers of the men yearly by the instructor of the St. John's Ambulance Brigade. All underground supervisors and about one-third of the men have received the Brigade certificate and gold button, as evidence of proficiency.

To help maintain interest in first aid, citations of good work are made in the safety inspector's daily report, interdepartmental first-aid contests are held, and two mine teams are annually entered in the Provincial and Dominion contests of the St. John's Ambulance Brigade. Last year the McIntyre teams won the McCrea First-aid Trophy for Ontario mines. First-aid equipment is kept available underground in case of accident.

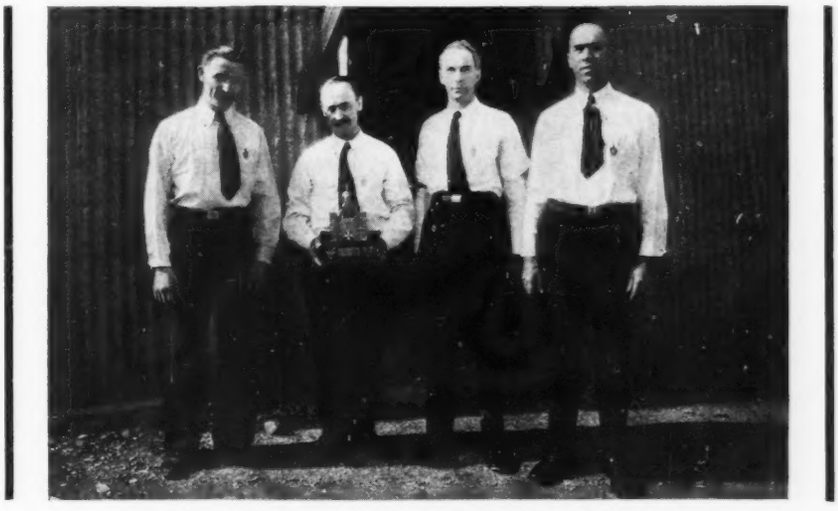
Fire-Prevention and Rescue Work—Fire prevention in the mine and training for fire fighting are given definite attention. The ore is not combustible, but the danger from the gases of even a small fire, in the confined spaces of the mine, is recognized.

Ethyl-mercaptan stench bombs are in place in the compressed air lines ready for the fire alarms. A schedule of alarms and the corresponding duties of officials has been worked out; fire doors are in place at each shaft station; standard fire hose with nozzles are connected to the water lines ready for use at each shaft station and at certain timbered areas; water for mining purposes is piped to each working place and is available for fire fighting. Two chemical engines on mine trucks are kept ready for use at Nos. 11 and 5 shaft houses. A continuous educational campaign and inspection is kept up—aimed at fire prevention.

The mine has available for rescue work and fire fighting: 1 LaFrance, double-40 gal. mine chemical engine with 200 ft. of hose at No. 11 shafthouse; 1 Bickle, 25-gal. copper tank chemical engine with 200 ft. of hose at No. 5 shafthouse; 12 All-Service gas masks and spare canisters; 12 M.S.A. Self-Rescuers; 1 M.S.A. carbon monoxide detector; and 2 H.H. inhalators.

The main rescue equipment for mines of the district is kept in a central Government Rescue Station, maintained with a permanent official in charge by the Ontario Compensation Board, through assessments on the mines. This equipment consists of the following items:

- 12 Sets McCaa, two-hour type, self-contained oxygen-breathing apparatus
- 12 All-Service gas masks
- 24 M.S.A. Self-Rescuers
- 2 M.S.A. carbon monoxide detectors
- 1 Pyrotannic acid carbon monoxide detector and air sampling accessories



First-aid team, winners of the Hon. Charles McCrea St. John's Trophy for Ontario mines, 1932

Men at No. 11 shaft ready to go underground on double-deck 60-man cage

2 Wolf flame safety lamps
12 Model "F" Edison electric cap lamps
1 M.S.A. high-pressure oxygen pump (hand-operated)
1 M.S.A. life line on reel
Accessories, such as goggles, tools, and essential repair parts.

Five mine rescue and fire-fighting crews of five men each are kept in training in the district, the men being chosen from the five chief mines of the area in numbers proportional to the number of men employed. Members are paid for training time and receive 25c. per day as a standby allowance.

Workmen's Accident Compensation—Compensation for injuries sustained in the course of employment and for silicosis, which is classed as an industrial disease, is handled by the Ontario Workmen's Compensation Board, which assesses each mine. The scale for such injuries is 66 per cent of wages lost, where lost time is at least one week, and complete medical and hospital care. Partial and complete permanent disabilities are compensated for on a "monthly pension for life" basis. Widows' allowances are $40 per month, with $10 per month for each child under 16. The rates which the mine pays to the board depend on the year's experience of the Ontario gold mines.

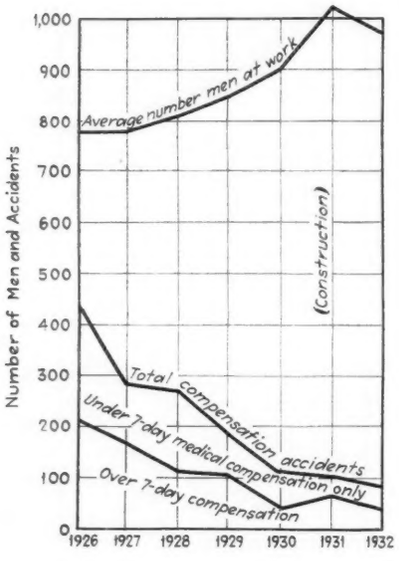


Fig. 1—Chart showing decrease in number of accidents despite increase in number of men working

The company's physician and his first-aid men record all dressings and sickness consultations. Each superintendent is responsible for reporting to the office accidents in his department. Accidents that result in loss of time or require more than first-aid are reported by the mine at once to the Ontario Compensation Board and to the local Ontario mine inspector. A record of all accidents is kept in the office.

Despite increased mechanization and larger tonnages from greater depths in Ontario gold mines, accident hazard has been decreasing because of the prevention efforts of officials of this property and other mines. This is reflected in compensation rates.

Accident Prevention—McIntyre was one of the first Ontario mines to have a safety organization and to devote attention to safe mining. The general manager and the superintendents and their staffs consider safety a major operating problem and have given it a definite place in design of plant and equipment and in planning and carrying out operations. It is recognized that if accident compensation costs, with their attendant losses resulting from interference with production, can be kept low, human suffering and employees' losses due to accidents will also be small.

Lost-time accidents among the general surface, mechanical, and mill crews are rare. The mill with 2,000 man-shifts per month has operated 358 days up to the present date (Oct. 30) without a lost-time accident.

In the mine, where 75 per cent of the men work, greater inherent dangers occur than on surface. The nature of the orebodies, particularly at depth, and the great number of working places in proportion to tonnage mined, make the natural hazards of mining at McIntyre more than at the average mine. The ground does not stand well, and considerable timber has to be used even in the cut-and-full stopes. Scaling of loose ground is a daily operation. The organized safety campaign has therefore been mostly concerned with the underground work.

Though suggestions concerning safety are welcomed by the safety inspector and mine officials, and often acted on, the mine superintendent, his assistant, the mine captains and shift bosses, without safety committees, assume responsibility for mine safety as an operating problem. Conferences of these men are held twice a month. These are an important feature in general mine cooperation as well as in getting safety results.

The Laws and Regulations of the Ontario Department of Mines, governing safe operation of mines, are the main rules of the mine, but an individual set of safety rules for the McIntyre has also recently been supplied to each employee.

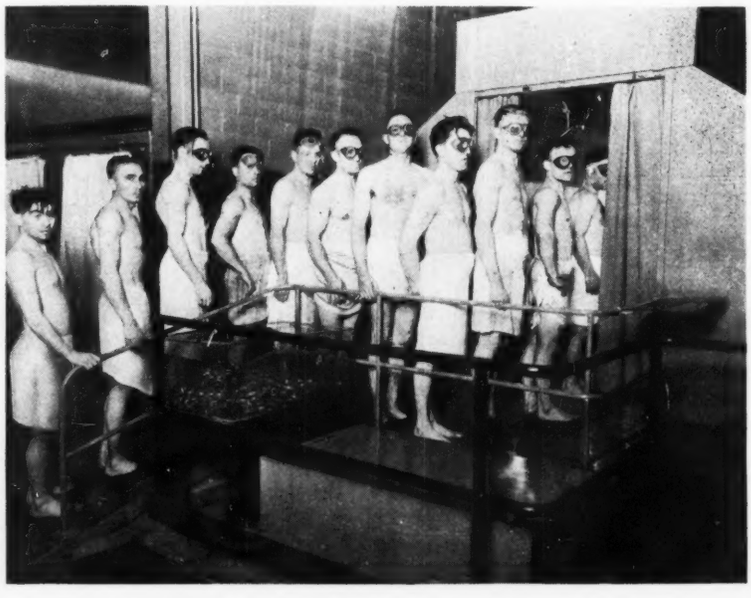
Safety bonuses of $75 are paid to shift bosses whose shifts go for three months without a lost-time accident or lost days from an accident of the previous quarter. In this, a lost-time accident is defined as one that causes a loss of more than two and a half working days. This time provision gives the boss a definite interest in looking after first aid and in preventing malingering. An average of 60 per cent of the shift bosses receive the quarterly safety bonus. One underground boss with 45 men in charge has had it for seven consecutive quarters.

Cooperation for Safety

The company is a member of the National Safety Council, the Ontario Safety League, and the Accident Prevention Association of Class V (mining and quarrying) Ontario Compensation.

Safety guards and devices are used and machinery and underground openings are guarded where possible. Protective clothing is in use, particularly in the mine. Fortunately the mine is cool, ranging from 40 deg. close to surface to 59 deg. at the 3,875-ft. level, so that clothing can be comfortably worn. Use of gloves, hard hats, safety-toe leather boots with safety rubber boots for a few wet places, goggles for sampling and sledging, Wilson narrow

Men entering solarium, in change house, for sun-lamp treatment



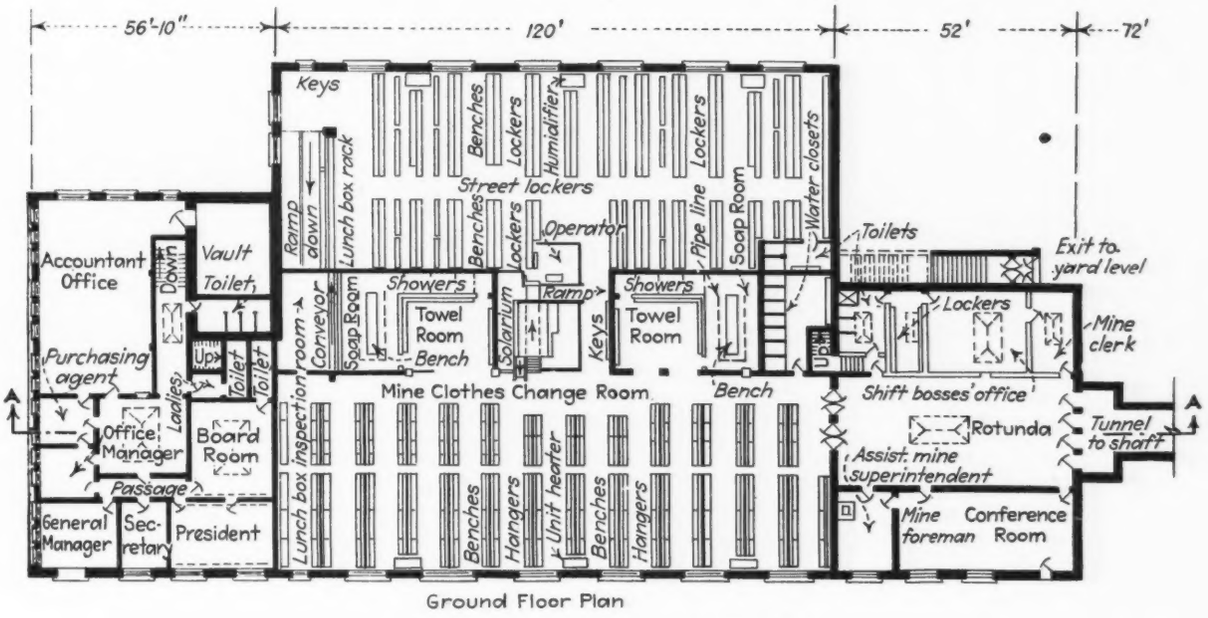


Fig. 2—Plan of change house and office building

bag-type respirators for short-time dusty work, and substantial ordinary overalls are all compulsory. Gloves are sold at cost, hats and respirators at one-third discount, goggles are free, and boots are handled by local merchants. The efficiency of this equipment has been fully demonstrated.

The safety campaign carried on for several years has definitely improved the accident rate. With 973 men employed through 1932, the number of compensation accidents was 89, of which only 41 resulted in over seven days' lost time with resultant compensation. The accident frequency rate was 33.6 per 1,000,000 hr. worked, and the severity rate, with the arbitrary time allowances as set by National Safety Council rules for all permanent disabilities, was 1.66 days lost per 1,000 hr. worked. No fatal accidents occurred last year and no man was hurt badly enough to prevent return to work. This year's record, unfortunately, will not be so good.

Gratifying decrease in the number of accidents, despite the increased number of men employed, is shown by the accompanying chart. Increase in the number of regular employees has been mostly underground. Previous to 1930, the labor turnover was heavy. Training such a high proportion of new men to safe production was a heavy task. Fortunately, the men now stay more steadily on the job. In 1932, it was possible to increase the underground working average by 100 men, or from 525 to 625, by hiring only 193 new men, of whom 35 were students or other temporary employees.

McIntyre change rooms are part of a central administration building erected when the new plant was centralized two years ago. This building, close to the main No. 11 shaft, contains the accounting and supervisory offices, assay laboratories, and change rooms for all employees. The change rooms for the underground men are unique. They were especially designed, with due regard for the psychology of the miner, by the McIntyre staff, to maintain the miners' health and general morale, and to facilitate that supervision necessary for efficient, low cost, safe-from-accident mining.

In marked contrast to the old odorous "dry," the rooms are commodious, clean, and attractive. Double change rooms are provided, one for mine clothes, the other for street clothes. Between are continuous shower bath and a solarium which every miner uses every day. The accompanying floor plan (Fig. 2) shows the layout.

The whole structure is substantially built with steel framework, and is as nearly fireproof as possible. In the lower basement, floors and walls are of solid concrete and the walls of the other two stories are of tile brick. The front wall is of red, pressed brick. The inside is plastered, and all rooms are attractively painted, well lighted and heated, free from draughts and kept in sanitary condition.

A tunnel provides direct access from No. 11 shaft, up which all the miners come, 60 at a time, on the double-deck cage. There is thus no exposure in damp clothes to outside cold or draughts or any opportunity for outside con-

Compressed air-water spray in a cut-and-fill stope

tacts. Automatic fire doors in the tunnel separate shaft and change house. A well-lighted rotunda at the change room end of the tunnel has, on each side, rooms for the underground supervisory staff. This gives adequate accommodation for planning of underground work, issuing of work orders, receiving of progress reports and that close supervision, instruction, and discipline which must be given those men who work in pairs in scattered, isolated places.

In the mine-clothes room, kept at this time at 76 deg., the miner after placing his lunch kit on a traveling belt to go through the inspection room, takes off all of his mine clothes and places them on his individual, numbered four-hook hanger, of which there are 860. Hanger and clothes are hoisted close to the ceiling by a rope, and by the next shift the clothes are dry and warm.

Modern Facilities

From this room the miner, unencumbered by clothes or lunch box, enters either of two duplicate shower bath rooms. In the soaping section, with overhead jet fixtures discharging a fine hot spray at the sides of the room, the bather applies a thorough lathering of soap and does most of his washing. As he passes along, continuously running water jets from each side of a 5-ft. passage give side showers of hot water, kept by thermostatic control at 105 deg., then farther on of medium water, and at the end a dash of cold water. A rub-down in the toweling room completes a pleasant, invigorating shower. The design, as may be inferred from Fig. 2, permits numbers of men to pass through easily and quickly.

From the shower rooms the miners pass by a traveling platform through a Burdick solarium for ultra-violet ray lamp treatment. The importance of this to these men, whose underground work the sun's rays do not reach, can be readily realized. The miners step from the solarium to the street-clothes room with skins tingling and with their tiredness gone.

Each miner has his own full-length locker for street clothes. Here all the men dress and proceed home, clean and rejuvenated by their baths and sun-lamp treatment. Lunch kits, after their trip through the inspection room, are received at the street-clothes room exit. The length of time taken by the individual miner, coming from the shaft, to report to his shift boss, change clothes, take showers and sun-lamp treatment, and punch time cards at the exit door is approximately 20 min. on the average.

These change rooms function well. General health of the miners has improved and there is less sickness and greater resistance to silicosis; accident experience has been encouraging, with 1932 establishing a low record in the number, severity and cost; high-grading of gold, with its suspicions and losses, has been prevented—a welcome relief to miners and supervisors. The building is a great aid in conducting mining in a planned, orderly manner. This expenditure of thought and money has proved itself to have been exceedingly worth while.

On the trail

Some cases of silicosis among the miners have been compensated by the Ontario Board. The condition has, however, been confined to comparatively few of those who have worked in the mine and thus been exposed to some silica dust. The disease here is only serious when complicated by tuberculosis. Investigation has been made at the mine into the statistics of the cases, the occurrence of the disease in other areas, its cause, and means of prevention.

In wet drilling, compressed air-water sprays, and ventilation are used in the campaign against dust underground. Experiments are being carried on with the view to precipitating mine dust chemically or electrically, and also in cloth filtering.

Dust surveys of mine air and of particular operation are made from time to time. Dust samples are taken by the circular konimeter. The microscope for counting the number of particles has a magnification of 160 diameters, the count being made on a dark ground illumination in the standard South African way, except that, the rock being different, the samples on the glass slides are here not acid-treated to remove solubles.

Counts are reported as the number of particles of dust under 5 microns in size in a cubic centimeter of air. Recently photographs of the konimeter samples taken through the microscope at 200 magnification have been used and have proved successful in demonstrating to the miners the dust content of air at particular places.

The miner's physical condition and the general health conditions of the mine affect the occurrence of silicosis. To increase his resistance to disease was one of the objectives for building the new change rooms. Further advances will be made in the fight against silicosis when the general community and the individual miner combat tuberculosis infection at all points away from the mine.

Health Factors With Depth—Mining at depth in the new No. 12 sub-shaft is not expected to develop unfavorable health conditions. Temperature will range from 58.5 deg. F. at the 3,875 level to 72 deg. F. at the 7,000 level. These are favorable temperatures. A separate, tightly lined air compartment of the shaft will make it possible to ventilate these workings adequately. Possible unfavorable effects on the miners due to change in air pressures on being hoisted from the 7,000-ft. level to the surface will be neutralized by the stage hoisting in the two shafts. Protection from surface exposure to cold is provided for by the new change rooms. Health conditions for work below the 4,000-ft. level will be at least as favorable as they are elsewhere in the mine.

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TRENDS AND DEVELOPMENTS IN THE MINING INDUSTRY

Mint Receipts and Gold-Mining Activity Increase Under New Monetary Policy

Reconstruction Finance Corporation Announces Purchase Rates Substantially Above Current World Prices—Payment Made With Notes Against Application Filed by the Producer

GOLD MINING—three months ago the "forgotten industry" under the New Deal—became, quite unexpectedly, on Oct. 22 the country's "most favored industry." This remarkable and rapid change in status was effected by President Roosevelt's radio announcement on that day of a new monetary policy, which, within a few days, resulted in the gold miner receiving more than the world price for his metal. The President, in the course of his announcement, said:

"Our dollar is now altogether too greatly influenced by the accidents of international trade, by the internal policies of other nations and by political disturbance in other continents. Therefore, the United States must take firmly in its own hands the control of the gold value of our dollar. This is necessary in order to prevent dollar disturbances from swinging us away from our ultimate goal, namely, the continued recovery of our commodity prices.

"As a further effective means to this end, I am going to establish a government market for gold in the United States. Therefore, under the clearly defined authority of existing law, I am authorizing the Reconstruction Finance Corporation to buy gold newly mined in the United States at prices to be determined from time to time after consultation with the Secretary of the Treasury and the President. Whenever necessary to the end in view, we shall also buy or sell gold in the world market."

Under the aforesaid authorization, subsequently set forth officially in an Executive Order dated Oct. 25, the Reconstruction Finance Corporation began, as of the same date, to announce prices at which it was prepared to buy newly mined gold. These prices throughout the last week of the month were well above the world price of the metal, and ranged from $31.36, the price established on Oct. 25, to $32.12, the price on Oct. 31. The margin by which the prices exceeded the world price during the period fluctuated from about 37c. on Oct. 25 to about $1.44 on Oct. 28, but on Oct. 30, following the announcement of the government's intention to buy gold abroad, this margin fell to about 44c. and then advanced to about $1.05 on Oct. 31.

The trend of the price that the gold miner has received for his product and that of the London price of gold are shown in the accompanying graph. From Sept. 8 to Oct. 24, inclusive, the curve for the United States gold quotation shows the variations in the "adjusted" world price as determined by the Secretary of the Treasury, at which rate the mints were authorized to purchase and the Federal Reserve Bank to sell newly mined gold. Sales of such gold in the foreign market under this arrangement totaled 346,529 oz. to the end of October, 89,175 oz. being sold in September and 257,354 oz. in October.

As shown in the accompanying table of world gold production, receipts of gold at U. S. mints increased substantially in September under the then prevailing favorable prices of the metal; that is, favorable compared with the previous $20.67 basis, at which level producers were paid prior to Sept. 8 for their metal at the mints.

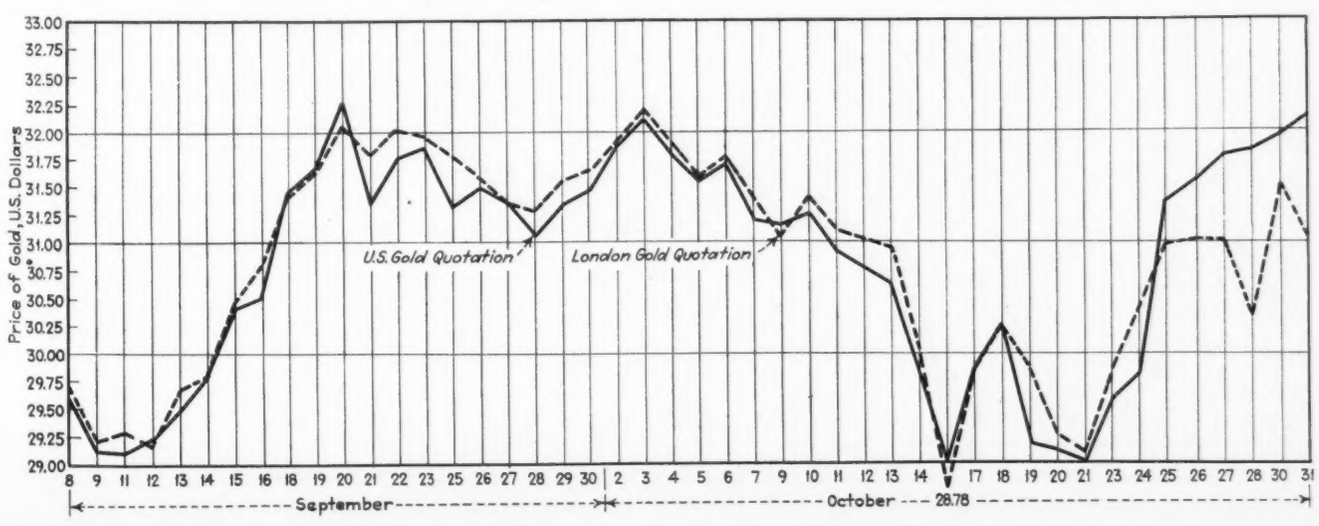
Since the new monetary policy began to function on Oct. 25, further increase has been reported in the flow of gold to the various mints of the country. On Oct. 29 the President announced the intention of the government to enter world markets to buy gold, the objective being the furtherance of the plan to control the gold value of the dollar.

The paragraph of the Executive Order dated Oct. 25 authorizing the Reconstruction Finance Corporation to buy and sell gold reads as follows:

"Section 3. The Reconstruction Finance Corporation is authorized, subject to such regulations as may from time to time be prescribed by the Secretary of the Treasury, to acquire gold which has been received on consignment by a United States Mint or Assay Office, and to hold, earmark for foreign accounts, export, or otherwise dispose of such gold."

Subsequent regulations issued Oct. 25 by the Secretary of the Treasury relative to the gold received on consignment by the mints and assay offices read as follows:

"Article 29. *Gold received on consignment.*—The United States Mints and Assay Offices under the conditions specified in this and the following Articles of these Regulations and subject to the appropriate regulations governing any United States Mint or Assay Office, will receive on consignment for delivery to the Reconstruction Finance Corporation gold which such Mint or Assay Office is satisfied has been recovered from natural deposits in the United States or any place subject to the jurisdiction thereof, *provided,* however, that no



gold shall be received under the provisions hereof which in the opinion of the Mint was held at any time in noncompliance with the Act of March 9, 1933, and the Executive Orders and Regulations issued thereunder, and provided further that no Mint or Assay Office shall receive on consignment any gold which in its opinion has theretofore entered into industrial or monetary use.

"Gold will be received in amounts of not less than two ounces of fine gold and in the following forms: bars, kings, buttons, retort sponge, lumps, grains, and dust, in their native state free from earth and stone, or nearly so.

"Consignments shall not contain less than 200 parts of gold in 1,000 by assay. In the case of gold forwarded to a Mint by mail or express, the original package will not be opened until an invoice of the description and weight of each such package shall have been received. When there is a material discrepancy between the actual and invoice weights of a consignment, further action with regard to it will be deferred pending communication with the consignor.

"Article 34. *Disposition of gold received on consignment.*—When, after a delivery of gold as provided in Article 29, the Mint is satisfied that the same may properly be accepted under the provisions of the Executive Order of October 25, 1933, and of these Regulations, and that the consignor has fully complied with the same, and after assay and receipt of Mint charges, it shall certify to the Federal Reserve bank in the district in which the Mint is located that it has available, in accordance with the Executive Order of October 25, 1933, for the account of the person by whom or on whose behalf the gold was consigned, the amount of gold shown by such assay. Upon receipt of information from the Federal Reserve banks that gold has been accepted by the Reconstruction Finance Corporation, the Mint shall dispose of such gold in accordance with instructions from the Reconstruction Finance Corporation or its agent."

The Reconstruction Finance Corporation announced on Oct. 26 that the notes which are to be issued for newly mined gold will mature on Feb. 1, 1934, and be sold to the gold owner at ¼ of 1 per cent per annum discount, to net the gold owner the quoted price from day to day at which gold will be accepted by the RFC.

Subscriptions for these notes may be made through the Federal Reserve Banks as fiscal agents for the RFC. For the time being the gold will be held for the account of the RFC by the Federal Reserve Banks, or at the mints.

The gold owner who wishes to buy notes, in the manner provided, will have no difficulty in converting his notes into cash at any bank that is a member of the Federal Reserve System for the quoted price for gold on the day that he subscribes for the RFC notes.

Stepping up of the gold price does not mean, according to Washington sources, that mining profits will increase proportionately. Mines will handle lower-grade ores. The increased price is certain to prompt demands for higher wages, which probably will be granted in many instances. The increase in the price of gold also means higher prices for supplies.

The feeling in official quarters is that gold production will not be increased as much as the uninformed probably will expect. Estimates are that the 1934 yield will probably not exceed that of 1933 by more than 10 per cent, but the new policy will mean that a greatly increased tonnage of ore will be treated.

The fact that gold producers must make an affidavit as to the origin of the gold, upon which federal authorities will base their purchases, means that the secret service of the Treasury Department will have the new duty of guarding against the purchase of foreign gold or gold other than that newly mined.

The Federal Trade Commission, which administers the new securities act, anticipates a flood of applications for the floating of securities of new gold-mining companies. Because of the new legislation the belief prevails that the public will be well protected against dishonest and ill-conceived promotions. Information already has reached federal authorities that the sale of stock in many gold-mining companies is being pressed actively in all parts of the country. The securities act is not retroactive and does not apply to issues brought out before its enactment.

World Production of Gold

(*American Bureau of Metal Statistics*)

In Fine Ounces

1933	July	August	September
United States (*a*)..	176,000	181,000	271,000
Canada.....	256,000	250,000	245,000
Mexico.....	56,000	58,000	(*f*) 58,000
Colombia.....	39,000	38,000	(*f*) 38,000
Other South America.....	39,000	40,000	44,000
British India (*e*)...	28,000	28,000	28,000
Japan (*e*).....	36,000	36,000	36,000
Queensland.....	7,000	8,000	(*f*) 8,000
Western Australia.	54,000	56,000	59,000
Other Australasia (*b*).....	36,000	36,000	36,000
South Africa.....	930,000	935,000	901,000
Belgian Congo....	22,000	22,000	21,000
Rhodesia.....	55,000	56,000	56,000
British West Africa	29,000	30,000	31,000
Russia (*c*).....	(*f*) 160,000	(*f*) 160,000	(*f*) 160,000
Elsewhere (*d*).....	(*f*) 75,000	(*f*) 75,000	(*f*) 80,000
Totals.....	1,998,000	2,009,000	2,072,000

(*a*) Includes Philippines. (*b*) Includes New Zealand and New Guinea. (*c*) Chiefly Siberia; 1933 estimated at average rate of 1932. (*d*) Includes West Indies, Central America, Europe, and Asiatic and African lands not separately reported; estimated. (*e*) Principal mines; nearly complete. (*f*) Conjectural.

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Gold-Mining Interests Form National Association

About 300 mine operators, mining engineers, and representatives of gold-mining interests met on Oct. 24 in San Francisco, Calif., to form a national association to protect both large and small gold-mining operators in the United States and possessions. Delegates from California, Nevada, Montana, Colorado, New Mexico, Washington, Utah, Oregon, Alaska, and the Philippines attended. At the speakers' table were George W. Starr, president of the Mining Association of California; J. C. Kemp van Ee, Jr., secretary of the association, and Charles L. Gilmore, mining attorney, of Sacramento.

The principal speaker was the Hon. H. L. Englebright. Mr. Englebright returned recently from Washington, D. C., where he and Western colleagues waged a fight in the interests of gold mining and for remission of the embargo on gold to permit the producers of the United States to sell their product at world prices. Mr. Englebright gave a brief account of his work in Washington, and explained the difficulties that confronted the Administration after declaration of the embargo on gold.

W. W. Bradley, State Mineralogist, urged gold producers to cooperate, and emphasized the need of a national association of gold producers. P. W. Racey, former superintendent of the Argonaut Gold Mining Company, presented the views and needs of the mine operator, and criticised the aloofness and unwillingness characteristic of the gold miner. Former U. S. Senator Tasker L. Oddie, of Nevada, described the battles the few representatives of the gold interests in Washington had to wage to protect the producers. In concluding, he paid tribute to Representative Englebright in behalf of the gold miners.

The conference voted to call the new organization the Gold Mining Association of America. The home office is to be in San Francisco. Interesting and lively discussion ensued after reading of the bylaws by Secretary Kemp van Ee, Jr. Suggestions and constructive criticism were offered by Julian Boyd, consulting engineer, Los Angeles; ex-Governor Campbell of Arizona; Dr. Henry M. Payne, of New Mexico; W. H. Metson, mining attorney, San Francisco, and representatives of the mining associations of Washington, the Southwest, Colorado, and others.

The new organization will be supported by a levy of 3c. an ounce on gold produced (the levy not to exceed $5,000 per annum per producer or individual) by operating companies, and 1c. a man per day employed by non-producing companies. Individual members will pay $12 a year. Following the adoption of the proposed bylaws the assembly unanimously elected George W. Starr, president, and J. C. Kemp van Ee, Jr., secretary-treasurer of the newly formed association. Members of the executive council are Cooley Butler, Los Angeles; Errol MacBoyle, Grass Valley; and Charles H. Segerstrom, Sonora.

Resolutions adopted provided that at the next session of Congress the National Securities Act be simplified so that it will be more applicable to primary financing; that steps be taken to expedite the payment on bullion sent to the Mint, and that the price of mining and milling supplies be investigated, particularly those controlled or manufactured by monopolies, for the purpose of obtaining lower prices.

Daily and Average Monthly Prices of Metals

October, 1933

United States Market

Oct.	Electrolytic Copper Domestic	Electrolytic Copper Export	Straits Tin New York	Lead New York	Lead St. Louis	Zinc St. Louis
2	8.775	7.975	48.875	4.50	4.35	4.750
3	8.775	8.025	48.450	4.50	4.35	4.750
4	8.775	7.950	48.625	4.50	4.35	4.750
5	8.775	7.925	48.950	4.50	4.35	4.750
6	8.525@8.775	7.900	48.875	4.50	4.35	4.750
7	8.525	7.900	48.500	4.50	4.35	4.750
9	8.525	7.725	48.250	4.50	4.30@4.35	4.750
10	8.275	7.625	48.250	4.35@4.50	4.35	4.750
11	7.775	7.525	47.875	4.50	4.35	4.750
12	Holiday	7.450	Holiday			
13	7.775	7.375	47.150	4.50	4.35	4.750
14	7.775	7.375	47.125	4.50	4.35	4.750
16	7.525	7.300	46.500	4.40	4.25	4.750
17	7.275	7.325	46.875	4.25	4.10	4.750
18	7.275	7.350	47.075	4.10	3.95@4.00	4.750
19	7.275	7.200	46.200	4.00	3.85@3.90	4.750
20	7.275	7.175	46.000	4.00	3.85@3.90	4.750
21	7.275@7.400	7.175	46.150	4.00	3.90	4.750
23	7.400@7.525	7.225	47.125	4.00	3.90	4.725
24	7.525@7.775	7.375	47.875	4.00	3.90	4.725
25	7.775@8.025	7.625	48.625	4.15	4.05	4.750
26	7.775@8.025	7.625	48.625	4.30	4.15	4.750
27	7.775@8.025	7.575	48.375	4.30	4.15	4.750
28	7.775@8.025	7.600	48.400	4.30	4.15	4.750
30	7.775@8.025	7.575	48.825	4.30	4.15	4.750
31	7.775	7.600	48.875	4.30	4.15	4.750
Av. for Month	7.950	7.557	47.858	4.313	4.176	4.748
Averages for Week						
4	8.733	7.904	48.325	4.500	4.350	4.750
11	8.421	7.767	46.450	4.488	4.346	4.750
18	7.525	7.363	46.945	4.350	4.205	4.750
25	7.483	7.296	46.996	4.025	3.917	4.742
Calendar Week Averages						
7	8.713	7.946	48.713	4.500	4.350	4.750
14	8.025	7.513	47.730	4.485	4.345	4.750
21	7.327	7.254	46.467	4.125	3.996	4.750
28	7.785	7.504	48.171	4.175	4.050	4.742

Silver, Gold, and Sterling Exchange New York and London

Oct.	Sterling Exchange "Checks"	Sterling Exchange "90-Day Demand"	Silver New York	Silver London	Gold London	Gold (a) United States
2	4.78000	4.81000	40.0000	18.5000	133s 5 d	$31.88
3	4.78250	4.81250	39.7500	18.3125	134s 8 d	32.12
4	4.76250	4.79250	39.8750	18.3125	134s ...	31.79
5	4.74000	4.77000	39.6250	18.5000	133s 8½d	31.55
6	4.74250	4.77250	39.6250	18.5000	133s 11 d	31.72
7	4.68625	4.71625	39.0000	18.4375	133s 4 d	31.20
9	4.68000	4.71500	38.7500	18.5000	133s 1 d	31.14
10	4.69000	4.72000	38.6250	18.3125	133s 9 d	31.26
11	4.65000	4.67250	38.0000	18.3125	133s 1½d	30.91
12	Holiday	Holiday	Holiday	18.3125	133s 5 d	Holiday
13	4.60000	4.62500	37.5000	18.3125	132s 10½d	30.62
14	4.55500	4.57500	36.3750	18.1875	131s 3 d	29.83
16	4.50500	4.53250	36.1250	18.1250	128s 6 d	29.00
17	4.55000	4.57500	36.2500	17.7500	130s 11½d	29.86
18	4.57000	4.60000	36.7500	18.0625	132s ½d	30.33
19	4.53000	4.55500	36.2500	18.0000	129s 11 d	29.18
20	4.53000	4.55500	36.6250	18.0000	129s ½d	29.13
21	4.51500	4.54000	36.8750	18.1875	128s 6 d	29.01
23	4.61500	4.64500	37.6250	18.1875	129s 2 d	29.59
24	4.66000	4.69000	37.5000	18.0625	128s 1 d	29.80
25	4.75000	4.78000	38.6250	18.0625	130s 1 d	(b) 31.35
26	4.73750	4.77750	38.6250	18.0625	130s 9½d	(b) 31.54
27	4.71000	4.75000	38.5000	18.0625	131s 2 d	(b) 31.76
28	4.70625	4.74625	38.8750	18.1875	129s 8 d	(b) 31.82
30	4.78750	4.82000	39.6250	18.2500	131s 2½d	(b) 31.96
31	4.76000	4.80000	39.3750	18.2500	130s 7 d	(b) 32.12
Av. for month	4.66380	.....	38.190	18.221	.....	.....
Averages for Week						
4	4.75500	.....	39.604	.....	.....	.....
11	4.69813	.....	38.938	.....	.....	.....
18	4.55600	.....	36.600	.....	.....	.....
25	4.60000	.....	37.250	.....	.....	.....

Calendar week averages: New York Silver Oct. 7, 39.646; Oct. 14, 37.85; Oct. 21, 36.479; Oct. 28, 38.292.

(a) U. S. Treasury's gold price, per ounce troy. (b) Reconstruction Finance Corporation's quotation for domestic newly mined gold.

London Market

Oct.	Copper Standard Spot	Copper Standard 3M	Copper Electrolytic Bid	Tin Spot	Tin 3M	Lead Spot	Lead 3M	Zinc Spot	Zinc 3M
2.....	35.5000	35.6875	38.7500	222.6250	222.5000	12.0625	12.3750	16.8750	17.1865
3.....	35.1875	35.3750	38.5000	221.1250	221.0000	11.9375	12.3125	16.6875	17.0625
4.....	34.8750	35.1250	38.2500	221.7500	221.7500	11.9375	12.3125	16.6250	17.0000
5.....	35.0000	35.2500	38.2500	224.7500	224.7500	12.0625	12.3750	16.6875	17.1250
6.....	34.9375	35.1250	38.2500	224.5000	224.5000	12.1250	12.4375	16.7500	17.1875
9.....	34.5625	34.7500	37.7500	224.3750	224.1250	11.8750	12.2500	16.7500	17.0625
10.....	33.8750	34.0625	37.2500	224.8750	224.8750	11.8750	12.2500	16.6875	17.0000
11.....	33.2500	33.4375	36.5000	223.8750	223.8750	11.8125	12.1875	16.5000	16.7500
12.....	33.1250	33.3125	36.0000	223.7500	223.7500	11.8750	12.1250	16.5000	16.7500
13.....	33.3125	33.5000	36.5000	223.8750	223.7500	11.8750	12.1250	16.5000	16.7500
16.....	33.8125	33.9375	37.0000	222.7500	222.7500	11.8750	12.1250	16.5625	16.8125
17.....	33.1875	33.3125	36.5000	221.8750	221.8750	11.6875	12.0000	16.4375	16.6875
18.....	33.1875	33.3750	36.5000	224.0000	223.8750	11.6875	12.0000	16.3125	16.5625
19.....	32.5000	32.6875	36.0000	221.8750	221.7500	11.4375	11.7500	15.8750	16.1875
20.....	32.6875	32.8750	36.0000	221.7500	221.7500	11.3750	11.6875	15.8125	16.1250
23.....	32.5000	32.6875	36.0000	222.2500	222.1250	11.2500	11.6250	15.5625	15.8750
24.....	32.9375	33.0625	36.0000	222.8750	222.7500	11.5000	11.8750	15.6875	16.0000
25.....	33.4375	33.5625	36.5000	223.8750	223.6250	11.9375	12.1875	16.0000	16.3125
26.....	33.6250	33.8125	37.0000	225.0000	224.8750	12.0625	12.3125	16.0000	16.3125
27.....	32.9375	33.0625	37.0000	224.5000	224.3750	11.9375	12.0625	16.0000	16.1875
30.....	33.1875	33.3125	36.5000	225.2500	225.1250	11.8750	12.0000	16.1250	16.2500
31.....	32.8125	32.9375	36.5000	224.5000	224.2500	11.6250	11.8750	15.8750	16.0000
Av. for month.	33.656	.....	36.977	223.455	.....	11.804	12.102	16.310	16.599

The United States quotations are our appraisal of the major markets for domestic consumption based on sales reported by producers and agencies. They are reduced to the basis of cash, New York or St. Louis, as noted. All prices of domestic class are in cents per pound.

Copper, lead and zinc quotations are based on sales for both prompt and future deliveries; tin quotations are for prompt delivery only.

Quotations for zinc are for ordinary Prime Western brands. Zinc in New York is now quoted at 0.37c. per pound above St. Louis, this being the freight differential. Contract prices for High-Grade zinc delivered in the East and Middle West are based on a minimum

Silver and Sterling Exchange

	New York		London Spot		Sterling Exchange	
	1932	1933	1932	1933	1932	1933
January	29.780	25.400	19.623	16.883	342.515	336.060
February	30.136	26.074	19.573	16.885	345.141	342.114
March	29.810	27.928	18.336	17.588	363.463	343.138
April	28.298	30.730	16.923	18.440	374.731	357.565
May	27.755	34.072	16.868	19.046	367.370	393.106
June	27.466	35.663	16.844	19.078	364.471	413.216
July	26.700	37.630	16.930	18.341	354.865	464.760
August	27.986	36.074	18.000	17.877	347.449	449.944
September	27.870	38.440	17.998	18.272	347.003	466.240
October	27.195	38.190	17.813	18.221	339.500	466.380
November	26.698	.....	18.099	.....	327.458	.....
December	25.010	.....	17.110	.....	327.683	.....
Year	27.892	.....	17.843	.....	350.137	.....

New York quotations, cents per ounce troy, 999 fine. London, pence per ounce, sterling silver, 925 fine. Sterling exchange in cents.

Copper

	F.O.B. Refinery — Electrolytic			London Spot			
	Domestic		Export	Standard		Electrolytic	
	1932	1933	1933	1932	1933	1932	1933
January	7.060	4.775	4.741	39.459	28.557	46.200	33.244
February	5.965	4.775	4.710	36.917	28.481	41.381	32.556
March	5.763	5.011	4.779	33.039	28.179	36.786	32.370
April	5.565	5.395	5.185	29.943	29.576	34.190	33.681
May	5.237	6.698	6.569	28.548	34.071	32.833	38.163
June	5.145	7.773	7.484	26.872	36.759	30.841	41.000
July	5.053	8.635	8.446	26.071	37.917	29.107	41.524
August	5.219	8.768	7.937	31.401	36.071	34.784	40.227
September	5.978	8.753	7.788	34.986	35.122	38.318	38.339
October	5.733	7.950	7.557	31.890	33.656	36.190	36.977
November	5.131	.....	.....	31.972	.....	36.568	.....
December	4.813	.....	.....	29.088	.....	34.344	.....
Year	5.555	.....	.....	31.682	.....	35.962	.....

New York quotations, cents per pound. London, pounds sterling per long ton.

Lead

	New York		St. Louis		London			
	1932	1933	1932	1933	1932 Spot	1932 3 Mos.	1933 Spot	1933 3 Mos.
January	3.750	3.000	3.550	2.875	15.084	15.128	10.458	10.833
February	3.712	3.000	3.499	2.875	14.560	14.571	10.431	10.719
March	3.150	3.146	2.993	3.021	12.345	12.634	10.609	10.821
April	3.000	3.260	2.900	3.135	11.223	11.503	10.872	11.122
May	3.000	3.654	2.900	3.525	10.673	11.036	12.095	12.372
June	2.993	4.173	2.896	4.023	9.608	9.898	13.280	13.571
July	2.747	4.452	2.611	4.303	9.818	10.152	13.411	13.613
August	3.235	4.500	3.093	4.350	11.349	11.588	12.182	12.457
September	3.465	4.500	3.315	4.350	13.122	13.349	11.932	12.229
October	3.052	4.313	2.939	4.176	11.958	12.170	11.804	12.102
November	3.050	.....	2.931	.....	12.071	12.395	.....	.....
December	3.000	.....	2.877	.....	11.144	11.541	.....	.....
Year	3.180	.....	3.042	.....	11.913	12.164	.....	.....

New York and St. Louis quotations, cents per pound. London, pounds sterling per long ton.

Tin

	New York		London	
	1932	1933	1932	1933
	Straits		Standard, Spot	
January	21.804	22.692	140.219	145.708
February	22.018	23.500	139.143	148.544
March	21.863	24.221	129.810	149.120
April	19.244	27.136	108.935	157.944
May	20.948	36.051	122.286	186.207
June	19.659	44.097	114.530	219.964
July	20.931	46.356	125.863	216.673
August	22.985	44.794	142.017	215.210
September	24.779	46.665	152.705	216.893
October	23.936	47.858	151.280	223.455
November	23.354	.....	153.585	.....
December	22.677	.....	149.806	.....
Year	22.017	.....	135.848	.....

New York quotations, cents per pound. London, pounds sterling per long ton.

Zinc

	St. Louis		London			
	1932	1933	1932 Spot	1932 3 Mos.	1933 Spot	1933 3 Mos.
January	3.011	3.018	14.416	14.834	14.381	14.595
February	2.817	2.666	13.872	14.289	13.866	14.119
March	2.787	2.987	12.616	13.024	14.647	14.674
April	2.725	3.298	11.670	11.958	14.951	15.208
May	2.532	3.805	12.432	12.682	15.505	15.660
June	2.777	4.348	11.548	11.866	16.988	16.774
July	2.537	4.878	11.592	11.967	17.795	17.789
August	2.758	4.916	13.594	13.844	16.869	17.031
September	3.322	4.699	15.455	15.616	16.810	17.042
October	3.027	4.748	14.869	15.140	16.310	16.599
November	3.094	.....	15.264	15.534	.....	.....
December	3.124	.....	15.209	15.347	.....	.....
Year	2.876	.....	13.545	13.842	.....	.....

St. Louis quotations, cents per pound. London, pounds sterling per long ton.

Cadmium and Aluminum

	Cadmium		Aluminum	
	1932	1933	1932	1933
January	55.000	55.000	23.300	23.300
February	55.000	55.000	23.300	23.300
March	55.000	55.000	23.300	23.300
April	55.000	55.000	23.300	23.300
May	55.000	55.000	23.300	23.300
June	55.000	55.000	23.300	23.300
July	55.000	55.000	23.300	23.300
August	55.000	55.000	23.300	23.300
September	55.000	55.000	23.300	23.300
October	55.000	55.000	23.300	23.300
November	55.000	.....	23.300	.....
December	55.000	.....	23.300	.....
Year	55.000	.....	23.300	.....

Aluminum in cents per pound, 99 per cent grade.
Cadmium, cents per pound.

Antimony, Quicksilver, and Platinum

	Antimony (a) New York		Quicksilver (b) New York		Platinum (c) New York	
	1932	1933	1932	1933	1932	1933
January	5.976	5.722	64.900	48.500	40.000	26.480
February	6.489	5.738	66.304	48.614	40.000	24.000
March	6.188	5.901	72.537	52.676	40.000	24.667
April	5.746	5.876	72.125	54.580	40.000	26.800
May	5.170	6.264	66.380	56.500	39.500	28.500
June	5.034	6.500	59.481	60.038	37.500	30.000
July	5.000	7.262	53.580	62.900	35.200	32.320
August	5.144	6.986	47.444	63.500	35.000	33.000
September	5.606	6.880	47.500	64.580	33.800	34.560
October	5.575	6.843	47.600	66.500	33.000	36.000
November	5.771	.....	48.750	.....	33.000	.....
December	5.400	.....	48.500	.....	30.462	.....
Year	5.592	.....	57.925	.....	36.455	.....

(a) Antimony quotations in cents per pound, for ordinary brands. (b) Quicksilver in dollars per flask of 76 lb. (c) Platinum in dollars per ounce troy.

Pig Iron

	Bessemer		Basic		No. 2 Foundry	
	1932	1933	1932	1933	1932	1933
January	16.00	14.50	15.00	14.00	15.50	14.50
February	15.68	14.50	14.68	14.00	15.18	14.50
March	15.50	14.50	14.50	14.00	15.00	14.50
April	15.50	14.50	14.50	14.00	15.00	14.50
May	15.50	15.42	14.50	14.42	15.00	14.92
June	14.85	16.00	14.35	15.00	14.75	15.50
July	14.56	16.50	14.00	15.50	14.50	16.00
August	14.50	17.00	14.00	16.00	14.50	16.50
September	14.50	18.00	14.00	17.00	14.50	17.50
October	14.50	18.00	14.00	17.00	14.50	17.50
November	14.50	.....	14.00	.....	14.50	.....
December	14.50	.....	14.00	.....	14.50	.....
Year	15.01	.....	14.29	.....	14.79	.....

Iron in dollars per long ton. F.o.b. Mahoning and Shenango Valley furnaces, freight to Pittsburgh, $1.89.

price basis of 4½c. per pound.

Quotations for lead reflect prices obtained for common lead, and do not include grades on which a premium is asked.

London prices for lead and zinc are the official prices for the first session of the London Metal Exchange; prices for copper and tin are the official closing buyers' prices. All are in pounds sterling per long ton (2,240 lb.).

New York silver quotations are as reported by Handy & Harman and are in cents per troy ounce of silver, 999 fine. London silver quotations are in pence per troy ounce of bar silver, 925 fine. Sterling prices represent noon market demand.